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J. W. RICHARDS, PH. D. *President*

E. F. ROEBER, PH. D. *Editor and Secretary*

C. E. WHITTLESEY. *Treasurer*

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Albany Meeting of the American Electrochemical Society.

In our last issue we have already called attention to the exceedingly interesting program of the convention to be held from April 30 to May 2 in Albany, Schenectady and Troy. The meeting of the first day will be held in Albany, that of the second day in Schenectady, that of the third day in Troy. This kaleidoscopic change of scene, together with the various visits and excursions which have been arranged, should prove most attractive. The program of papers to be presented, as published on page 176 of this issue, is as full and diversified as ever and we note with pleasure that there are three lectures of great general interest on the program, the speakers being Dr. Bancroft, Professor Burgess and Dr. Steinmetz.

Plating Iron with Zinc

Iron and steel surfaces are coated with zinc, tin or paint principally for the purpose of providing protection against corrosion. The usual term "galvanized iron" for iron coated with zinc is unfortunate. There is no galvanic action *per se* in iron coated with zinc. Only if by chance or accident part of the iron surface is free from zinc and exposed to the atmosphere, then this part of iron, together with the surrounding zinc and the atmospheric moisture forms a short-circuited galvanic cell and there ensues an electric current from the zinc through the moisture into the iron. The zinc being the anode dissolves, while the iron as cathode is not attacked on account of the essentially reducing nature of every cathodic reaction. If there was any iron oxide present instead of metallic iron, it would be reduced to the metallic state by the current. This is the rationale of the protection of iron from rust by means of a zinc coating.

* * *

The interesting article in this number from the pen of Mr. Cowper-Coles, who is unquestionably one of the most competent men to discuss this subject, gives details of, and comparisons between, four different methods of coating iron with zinc. The first is the old and most generally employed method of dipping in molten zinc; while it may be in some respects less expensive than the other methods, as pointed out by Mr. Cowper-Coles, yet the "hot-galvanizing" method enables the manufacturer to push the output of his plant to the limit, which fact accounts for the favor which it has always found with our manufacturers. The second method is electroplating of iron surfaces with zinc. As has been pointed out by Prof. C. F. Burgess formerly in this journal, this method gives a decidedly superior and more effective coating. Moreover, much has been done in recent years to facilitate the operation by invention of labor-saving and fool-proof machinery. Both the molten zinc process and the electrolytic process can be carried out now in a practically continuous operation. The third and fourth processes, "sherardizing" and "cowperizing," are comparatively

recent inventions of Mr. Cowper-Coles, and are not yet used commercially in this country, though we may expect their arrival at an early date, since both are not only very interesting, but have distinct advantages for certain work. "Cowperizing" is essentially treatment in an atmosphere of zinc vapor. With respect to "sherardizing," Mr. Cowper-Coles emphasizes that the temperature employed is below the evaporating point of zinc (solid zinc). But it appears that the temperature is above the point at which zinc dust, as employed in this method, begins to vaporize. It would, therefore, seem that sherardizing is also to some extent at least a vapor process. The notes on the applicability of sherardizing for decorating and artistic purposes emphasize a point which is already clearly evident. There is little likelihood that any one of the four methods will supersede the others. Each of these will most likely find a field of its own to which it is specially adapted, and all of these together will tend to increase the quantities of iron and steel coated with zinc and to create new purposes for which this is being done.

Ore Dressing

The astounding growth which the art of ore dressing has shown in recent years has been rendered possible by the happy way in which scientific principles have been applied to casual observations made in every-day practice. Ore dressing is a mechanical process with respect to the result, which is separation without chemical change. But the means employed are no longer purely mechanical; they depend no longer on differences of specific gravity only, but use is also made of differences in magnetic permeability, in electric conductivity and in susceptibility to the lifting action of gas bubbles with or without oil. Mr. W. G. Swart's presidential address before the Western Association of Technical Chemists and Metallurgists, printed on another page of this issue, gives a particularly suggestive and concise review of the present situation. We have added some further notes on acid flotation and on the new Elmore oil vacuum process. The latter illustrates well the complexity of many of our modern methods. It is to some extent a combination of a "grease process" with ordinary flotation. Concerning the original discovery of grease processes an anecdote is told which is characteristic whether true or not. At the Kimberley diamond mines a boy was eating his sandwich and a piece of his lunch fell on the traveling belt on which the diamond-bearing clay passed. The belt was thus smeared with butter. The boy was observative enough to note that where the butter was there also were the diamonds. He made easy money in letting his employers into his secret. What Elmore now does is to grease the metallic particles in order to make them more susceptible of attaching gas bubbles; he produces the gas bubbles in a similar way as in ordinary flotation; he increases their lifting effect by producing a vacuum above the liquid; and he makes the whole process continuous.

The Power House of a Large Copper Refinery.

For very good reasons, which we need not discuss at present, the bulk of the large electrolytic copper refining industry of this country is located near New York City. There are the Nichols Copper Works, on Long Island, the American Smelting & Refining Company and the Raritan Copper Works, at

Perth Amboy, N. J., the U. S. Metals Refining Company, in Chrome, N. J., and the Balbach Smelting & Refining Company, in Newark, N. J. We have repeatedly referred to the very important additions which have been made to these works in recent years and which have resulted in doubling the aggregate capacity in about five years. It is with pleasure, therefore, that we publish in this issue a full description by Mr. F. D. Easterbrooks of the new power house of the Raritan Copper Works. It will be found exceedingly interesting in many respects from a constructional point of view, the drawings being particularly instructive. The power house of an electrolytic refinery is a proposition quite different from the ordinary large power house for electric light, power and traction. The characteristic feature of the electrolytic power house is the absolutely uniform load curve throughout the full 24 hours a day. This permits the production of electric energy at a reasonably low rate even with steam power, which is a very important matter, in view of the fact that the power cost is 40 per cent of the whole tank-house cost of copper refining from anode to cathode under present conditions. Figured on the basis of the total cost of refining copper from pig to wire, the power cost is estimated to amount to approximately 22 to 24 per cent of the total cost. These figures refer to the usual multiple system of refining.

Achievements and Technical Possibilities of Photochemistry.

Treated and looked upon in many respects as a veritable step-child of chemistry, photochemistry can boast of one single big technical achievement. Photography is a great art, especially now with the rapid strides made by professionals and amateurs alike in many directions, color photography being not the least interesting development. Yet, when we come down to the rationale of many of the reactions which we employ, we must honestly confess that they are full of mysteries. It is, however, an encouraging sign of the times that just at present we are witnessing a revival of photochemical research. Thus, the German Bunsen Society will have at its coming Vienna meeting a symposium of not less than eight papers on photochemical subjects, while the American Electrochemical Society will listen, at its Albany meeting, to a lecture of Dr. W. D. Bancroft on the electrochemistry of light. This manifestation of renewed interest in an old branch of chemical science is natural enough. The field is wide and almost limitless, and very, very little is known or firmly established. The harvest which will be reaped by the master mind who brings about order out of chaos, will be worth while. Moreover, the subject is not simply one of scientific or theoretical interest. As we intend to show presently, the photochemical reactions which Nature produces indicate technical and commercial possibilities of utmost fundamental importance for human civilization.

* * *

It is not without reason that the present revival of photochemical research is chiefly due to the endeavor of electrochemists. There is indisputable evidence that there must be a close interrelation between photochemistry and electrochemistry. We know that light rays, heat rays, the Hertzian rays employed in wireless telegraphy are all special cases of one and the same

kind of waves—electromagnetic waves. They all obey exactly the same typical laws and they only differ in one respect—the wave-length. Naturally the wave-length is important, and the only fundamental photochemical law which we know with some certainty to be true, states that only light of a color or wave-length can produce a photochemical reaction in a system, if the latter absorbs this special color or wave-length. This selective absorption of light of a certain wave-length is a very peculiar phenomenon. The preference of one wave-length over all others is often exhibited in a peculiar way by a substance, whatever its quantity or external geometrical form, whatever the temperature and the physical state, whether solid or liquid, or solution, etc. It would seem that something within the substance is “tuned” to the special wave-length and on the basis of the electronic theory we can devise interesting and ingenious pictures or models, though they are still too vague to be made much use of. But so much seems evident that the selective absorption of light and the corresponding photochemical processes are phenomena related in some way to electromagnetic resonance. Further photochemical reactions are in general oxidation and reduction processes and from the analogy with electrolytic processes we can conclude that we have here to do with exchanges of ions or ionic charges or electrons. But these are only general indications, though undoubtedly suggestive of some underlying truth. When we come down, however, to a special reaction, we are all at sea. Even with a reaction, like the change of oxygen into ozone by the silent electric discharge, which has been studied extensively and thoroughly, and is employed commercially, we do not know with certainty whether it is a specific electric effect or a photochemical reaction due to ultra-violet light or both. We do not even know whether we put the question correctly in asking whether it is an electric or a photochemical effect.

* * *

The technical possibilities of photochemical reactions are clearly indicated by the processes performed by Nature. The eye is a photochemical instrument, a sort of actinometer. The effects brought about by sunlight on our earth are to an important extent photochemical. On a small scale we have learned to make use of the photochemical effect of sunlight, as in bleaching, while in the destruction of certain paints we have an undesired effect of sunlight. But one of the most important photochemical effects of sunlight—the dissociation of carbon dioxide in green, live plants with the resulting regeneration of the oxygen content in the atmosphere—is a benefit of immense importance which we derive from sunlight without doing anything for it. We do not understand it. We cannot imitate it. G. Kirchhoff has once said that the object of theoretical physics (and, we may add, chemistry) is to “describe” the facts in as complete and concise a manner as possible. Kirchhoff’s meaning of a “description” cannot be misunderstood since his statement was made in the introduction of his lectures on analytical mechanics, and his method of a complete and concise “description” of mechanical processes consists in the use of differential equations. They describe exactly and concisely what will happen if the starting conditions are given. They describe in the same way what the conditions are with which we must start in order to get a desired result. When we are so far in our understanding of the laws of

Nature within a limited field, one of the chief difficulties is removed which the technical man and engineer has to meet. For we may say that it is the object of the engineer to imitate Nature in as concise and cheap a way as possible. To be able to do this, he must understand how Nature acts. In photochemistry we are still far from this point. We can do nothing at present but experiment and keep on experimenting and keep our eyes and our mind open in observing Nature. It is even quite possible that our present experimental methods of chemistry are too “barbarously destructive” to imitate those subtle, delicate reactions which Nature produces. As Dr. L. Baekeland recently remarked, “we might just as well try to imitate the melodious music of a Gounod by firing some dynamite cartridges between the delicate strings of a piano.”

* * *

Yet, the difficulties should not discourage us. We should look at the goal. Let us come back again to the work performed by the sun. We will not speak of the heat the sun gives us, nor of the action of the sun in the evaporation of water from the oceans and the resulting cyclic movement of water passing through evaporation and condensation and placing at our disposal the kinetic energy of rivers and waterfalls. Nor shall we speak of the sunlight of ages ago stored up in our coal deposits. Let us consider simply the one reaction, already referred to, of dissociation of carbon dioxide in green live plants, under the action of sunlight. What happens is this: In all combustion processes, in the commercial burning of fuel, as well as in the respiration process of animals, oxygen from the atmosphere is consumed. The atmosphere gets poorer in oxygen and richer in carbon dioxide. But the green live plants absorb from the atmosphere the carbon dioxide and under the influence of sunlight perform the reverse reaction, the decomposition of the carbon dioxide into carbon (which builds up the tissues of the plants) and oxygen which passes off into the atmosphere. Thus the composition of the atmosphere is kept materially constant. Without the photochemical reaction due to sunlight this would not be the case. It is useful to look at this matter in a “sordid” commercial way. What is this photochemical reaction due to sunlight worth to us in dollars and cents? Prof. R. Luther estimates that the artificial production of the same quantity of oxygen by dissociation of carbon dioxide—as is performed by green vegetation on the earth under the influence of light—would require daily at least 15,000,000,000 hp-hours. If we estimate the cost of one hp-hour as one-fourth cent only, this cost amounts to over \$37,000,000 every day. That is the daily value of a single photochemical reaction of the sun on our earth. It is a pious hope of the illuminating engineers that some day they will be able to imitate the glow worm, which is looked upon as the ideal source of light. Photochemists may with even greater justification strive towards the goal of imitating Nature in its photochemical reactions. When this goal is reached, the effect on human civilization cannot be imagined. Our present lack of ability of carrying out photochemical reactions at will under the influence of sunlight, which is at our disposal in unlimited amount, represents a waste, qualitatively just like the waste of non-developed water powers or non-utilized blast-furnace or coke-oven gases, but a waste of immensely and uncomparatively greater importance.

Albany Meeting of the American Electrochemical Society.

The 7th annual and 13th general meeting of the American Electrochemical Society will be held on Thursday, April 30, Friday, May 1 and Saturday, May 2, in Albany, Troy and Schenectady.

The headquarters will be at the Hotel Ten Eyck, in Albany, N. Y. Members will please register and obtain badges upon their arrival in the lobby of the Hotel Ten Eyck.

Thursday, April 30. The morning session, beginning at 10:30 a. m., will be held in the ballroom of the Hotel Ten Eyck, where the following papers will be read and discussed:

The Construction and Accurate Measurement of Resistances of the Order of 10,000 Megohms. H. L. BRONSON.

The Absorption of the Radioactive Emanations by Coconut Charcoal. R. W. BOYLE.

Industrial Applications of Aluminium. E. BLOUGH.

The Nature of the Surface Film formed on Aluminium and Magnesium and the Aluminium-Magnesium Cell. H. T. BARNES and G. W. SHEARER.

Mercury Cathodes in Nitric Acid Solution. J. A. WILKINSON.

Copper Anodes in Chloride Solutions. PAUL DUSHMAN.

Solubility Determinations in Aqueous Alcoholic Solutions. A. SEIDELL.

The Potential of the Nickel Electrode. E. P. SCHOCH.

The annual business meeting will be held in the afternoon, also in the ballroom of the Hotel Ten Eyck, at 2 o'clock, where the report of the board of directors will be presented.

The retiring president, Prof. C. F. BURGESS, will then present his presidential address on the Corrosion of Iron from the Electrochemical Standpoint. This address will be open for discussion.

After this address the following papers will be read and discussed:

Electrochemical Corrosion of the Rochester Steel Conduit. RICHARD H. GAINES.

The Alloying of Iron and Calcium. A. HIRSCH and J. ASHTON.

The Preparation of Calcium Alloys for Aluminothermic Work. O. P. WATTS and J. M. BRECKENRIDGE.

Distillation of Turpentine by Electricity. F. T. SNYDER.

Water Power in the Adirondacks and Forest Preservation. E. R. TAYLOR.

Power for Electrochemical Industries. J. MEYER.

In the evening, at 8 o'clock, Dr. W. D. BANCROFT will deliver a lecture on the Electrochemistry of Light. This will be open for discussion.

After the lecture there will be an informal gathering in the café of the hotel.

Friday, May 1. There will be only one meeting in the morning. This will be held in the lecture hall of Union University, Schenectady. Electric cars will leave the Hotel Ten Eyck, Albany, at 9 a. m., going direct to the place of meeting. The meeting will be called to order at 10 a. m., when Dr. C. P. STEINMETZ will deliver a lecture on Kinetic Molecular Energy.

The following papers will then be read and discussed:

The Heat of Vaporization of Metals and Other Substances. JOS. W. RICHARDS.

Robert Hare's Electric Furnace. C. A. DOREMUS.

Electrochemical Patents. A. B. MARVIN, Jr.

The Electrical Testing of Iron during Annealing. E. E. F. CREIGHTON.

The Synthesis of Hydrocyanic Acid in the Electric Furnace. R. S. HUTTON and E. W. SMITH.

At 1:15 lunch will be served in the restaurant (building No. 80) of the General Electric Co.'s works. The society will be the guests of the company.

The afternoon will be devoted to an inspection of the General Electric Co.'s works. The party will be divided into

groups, which will be conducted by members of the staff of the company, and three hours are available to spend in these world-renowned works.

The train leaves Schenectady at 5:45 for Albany, where an informal dinner will be taken at the Hotel Ten Eyck at 7:30. It is hoped that all members and guests in attendance will be present.

Saturday, May 2. There will be a meeting in the morning, which will be held in the lecture room of the Rensselaer Polytechnic Institute in Troy. The meeting will begin at 9:30 and the following papers will be read and discussed:

Conduction in Electrolytes. L. KAHLBERG.

The Passage of Direct Current through Electrolytes without the Use of Electrodes. CARL HERING.

Reversed Electrolysis. J. W. TURRENTINE.

The Conversion of Iron Pyrites, FeS_2 into a Magnetic Form. O. L. KOWALKE.

The Electrolytic Refining of Iron. E. F. KERN.

Electrolytic Corrosion of Aluminium Bronze. W. S. ROWLAND.

Further Researches on the Electrolytic Corrosion of the Bronzes. A. T. LINCOLN.

Problems in the Manufacture of Dry Cells. C. F. BURGESS.

Lunch will be taken at the Rensselaer Inn at 1 p. m. The afternoon will be devoted to visits to points of interest in Troy, especially to Gurley's factory for making scientific instruments, the Burden Iron Works, the Earl and Wilson linen factories, the Watervliet Arsenal (open 1 to 4 p. m.), the Duncan Paper Mill of the West Virginia Paper Co., at Mechanicville. (The Hargreaves-Bird electrolytic chlorine plant of this company will not, however, be open to inspection.)

Dr. W. R. Whitney is chairman and Dr. C. F. Lindsay is secretary of the local committee.

Bunsen Society.

The annual meeting of the German Bunsen Society will be held in Vienna from May 29 to 31.

There will be a symposium of papers on photochemistry, by Prof. Luther, Dr. Trautz, Dr. Byk, Prof. Stobbe, Prof. Schaum, Dr. Scheffer, Colonel von Hübl and Prof. Wiesner.

Among other papers to be presented we notice the following: Dr. Hans Goldschmidt on new thermit reactions; Prof. Walter Nernst on the theory of the electric irritation of nerves; Dr. Bechhold on the question whether dyeing is a chemical or a physical process, etc.

Notes on Electrochemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

Metallurgical Research at the National Physical Laboratory During 1907.

The work undertaken on behalf of the Alloys Research Committee of the Institution of Mechanical Engineers embraces (1) further researches on the alloys of copper and aluminium, in continuation of the eighth report by Dr. Carpenter and Mr. Edwards, and (2) the investigation of other series of aluminium alloys.

Copper Aluminium Alloys: Tensile tests at temperatures up to 500° C. have been carried out on specimens turned from rolled bars containing 9.90 and 6.73 per cent. aluminium, respectively, in the electrically-heated tube furnace described in the report for 1906. In the case of the alloy containing 9.9 per cent. aluminium there was steady decrease of tenacity as temperature increased. Strength was fairly maintained up to 350° C., but above 400° C. weakness developed rapidly. The elongations were rather erratic, but indicated increased elongation with rising temperature. With the alloy containing 6.73 per cent. aluminium, tensile strength fell regularly with in-

creasing temperature, and more rapidly above 400° C., but in contrast with the first alloy, this showed a marked and regular decrease of elongation with increase of temperature. This is in accordance with these alloys having been placed in different groups in the 1906 report.

Sea-water corrosion tests are being carried out at Portsmouth Dockyard on specimens of copper-aluminium alloys, naval brass and Muntz's metal. After machining, some specimens were framed in teak and some were bolted onto mild steel plates. The first are free from galvanic action; the second are so disposed that action occurs only with the steel on which each specimen is secured. As it appears possible that the relative position of the alloys will not be the same on a prolonged test as on a short immersion, these tests will be continued for some time.

Further Series of Aluminium Alloys: Investigation of ternary alloys of aluminium with copper and one other element, e. g., manganese, zinc and nickel, has been undertaken, and a preliminary exploration of the mechanical properties of the manganese series has been nearly completed. The results cannot be stated at present, except that the mechanical properties of copper-rich alloys of aluminium and copper can probably be decidedly improved by additions of small proportions of manganese. The difficulties incident to the use of approximately pure manganese have led to experiments with a view to using a definite copper-manganese alloy obtainable commercially. The micro-structure of the preliminary alloys has been studied, but their thermal study by means of cooling curves, quenching, etc., is reserved until the constitution of the alloys of aluminium and manganese shall be better understood. A large number of aluminium-manganese alloys have been prepared for study of freezing points and cooling curves, but difficulty has arisen from the fact that the manganese available is only 97 to 98 per cent. pure, aluminium, silicon, iron and carbon being present. As it is possible that certain of the critical points of the alloys near the manganese end of the series may be wholly or partly due to these impurities, the preparation of a pure manganese-aluminium alloy containing as high a proportion of manganese as possible has been attempted, but the requisite degree of purity was not attained, although better results are anticipated from the use of purer aluminium. By means of such an alloy to which successive additions of aluminium may be made to obtain alloys with less manganese, the more important observations on the freezing and cooling curves will be repeated, so that for these members of the series the influence of impurities will be eliminated.

For alloys ranging from pure manganese down to the specially prepared pure alloy, this elimination of impurities will be difficult, and the determination of the freezing point of pure manganese, especially, must be carried out by other methods than those adopted for the ordinary metals and alloys. It is noteworthy that alloys lying near the manganese end of the series attack porcelain and fire-clay very rapidly, and thermo-junction protectors made of these substances are destroyed in a few minutes when immersed in the fluid metal. Tubes of pure magnesia have, therefore, been used, but they cannot be withdrawn from the molten metal without fracture. The study of the aluminium-manganese alloys already promises interesting results. Those containing between 65 and 30 per cent. of manganese are remarkable for their disintegrating spontaneously, and more or less rapidly, from the solid cast state into fine crystalline powder. The nature of the change causing this phenomenon is under investigation. The micro-structure of these alloys has also been studied, both when slowly cooled and when quenched from definite temperatures suggested by the cooling curves. The method of quenching consists in heating the specimen in a tube of vitreous silica from which the air has been pumped as completely as possible and which passes through the porcelain tube of an electric tube furnace without contact. Water is driven by atmospheric pressure into the exhausted tube, and the rapidity of cooling in this way appears to be considerably greater than that obtained by simple immersion of the

specimen in water, and approximates to that obtained by vigorous agitation of a specimen in cold water. The silica tubes appear to be unaffected by this treatment up to higher temperatures than 800° C.

Cooling Curve Research: The results obtained have been embodied in a paper read before the Physical Society of London, on Jan. 24, 1908. The general conclusion arrived at is that both the differential method and the direct method (the observation of time and temperature during the cooling plotted as inverse-rate curves) are capable of yielding delicate and accurate curves, but that the direct method requires much more delicate apparatus than the differential to obtain the same degree of accuracy. The effect of various rates of cooling, varying from one hour and three-quarters to twelve minutes for a range of 400° C., upon the form of the differential cooling curves has been tried, and it was found that these as ordinarily plotted were difficult to interpret when the rates of cooling were rapid, owing probably to the differences of temperature set up between the two bodies under comparison becoming great as compared with their rate of change at or near recalescence points. A "derived differential" curve, representing the differential coefficient of the ordinary differential curve with respect to temperature was, therefore, plotted. It bears to the ordinary differential curve a similar relation to that existing between the simple "time temperature" curve and the "inverse-rate" curve, and it greatly facilitates correct interpretation—the curves thus obtained for widely different rates of cooling showing a remarkable agreement. It has further been shown that the interpretation of the indications of cooling curves in terms of the quantity of heat evolved at a given recalescence is only possible, even to a rough approximation, when the rate of cooling of the body under observation has remained constant during the various recalescences to be compared. The experimental work of this research revealed a well-marked recalescence in crystalline silica at 580° C. Recalescence was observed in an electric-resistance furnace in which blank differential curves for a pair of platinum cylinders were determined. It was traced to the quartz packing of the furnace, and was found to exist in all the porcelains containing free silica, but not in those containing an excess of alumina, in which there was no free silica. Silver sand gave the recalescence at 580° C. in a very marked manner, both on heating and cooling, while amorphous silica (precipitated) and vitreous silica (fused quartz) were entirely free from reaction. Consequently the recalescence appears to be due to a polymorphic change in the crystalline silica; and there is a probability that a series of recalescence points at and just below 600° C. shown in cooling curves of steel published by Roberts-Austen, Carpenter and Keeling, and others should be referred to the silicious materials of the furnaces and not to the steel.

Eutectics Research: The lead-tin eutectic was prepared in a state approaching purity without difficulty, and its structure and behavior studied in detail. It was found that the view of Roberts-Austen and others that tin was only very slightly soluble in solid lead, was due to insufficient time having been allowed for the alloy to attain a condition of equilibrium. The alloys appear to require exposure to a temperature of about 175° C. for several weeks to attain complete homogeneity in the solid lead-tin solution. The exact limit of solid solubility at this end of the series has not been determined, but probably lies near 17 per cent. of tin. There is a well-marked recalescence in alloys lying between 5 and 40 per cent. of tin. In alloys between 18 and 40 per cent. this occurs on cooling at a constant temperature of 145° C., but below 18 per cent. the temperature of recalescence on cooling falls rapidly until at 5 per cent. it lies very near the ordinary temperature. For polishing, very soft alloys rich in lead oxide of chromium, obtained by calcination of ammonium bichromate and subsequent levigation in slightly ammoniacal water, are used, as recommended by Le Chatelier. The Zeiss apparatus for obtaining photo-micrographs by ultra-violet light has been installed. It consists of a

microscope and vertical camera so placed that the camera can be readily swung aside. The microscope has objectives of 6 mm and 1.7 mm focus of vitreous silica, while the eye-pieces and illuminating train are of rock-crystal. The light is derived from a spark between magnesium or cadmium electrodes, excited by means of a large induction coil with a mechanical interrupter and a battery of seven Leyden jars in parallel. This arrangement gives a short, thick spark with a large proportion of ultra-violet light of wave-length $.275 \mu$. In the apparatus of Messrs. Zeiss the light passes through a train of two 60° quartz prisms, and in the arrangement first tried in the laboratory the spectrum image of the spark was focussed upon the aperture of the vertical illuminator. In the apparatus devised by Dr. Koehler the image formed by the microscope is observed and focussed approximately by means of a searcher eye-piece provided with a screen of fluorescent uranium glass and a viewing eye-piece, and when the image appears in focus in the searcher it should also be in focus on the screen at a definite camera extension. Focussing was found to be very difficult.

Some sharp photographs with a magnification of about 400 diameters were obtained after much trouble, with the 6 mm objective, but with the more powerful lens and the high magnification which should constitute the principal value of this apparatus no satisfactory results could be obtained. Monochromatic blue light was then tried for finding the best arrangement of illumination for the avoidance of internal reflections, finding the required spot on the specimen, and finding the approximate focus. A small arc lamp gives a spectrum by means of a small optical bench provided with a prism and Thorp grating, and the spectrum is brought to a focus on a screen in which a small circular aperture allows a pencil of light of the desired wave-length to pass towards the aperture of the microscope. With due adjustment and further concentration of the light by a short-focus condenser just outside the illuminator aperture, an even, brilliant illumination of the field of the quartz objectives can be obtained, and an image which, with moderate eye-piece magnification, leaves nothing to be desired as regards definition. In the final arrangement it is intended to adjust matters so that by the insertion or removal of a small mirror the beam of light from the arc lamp can be instantly replaced by a beam of ultra-violet light from the spark-gap. It is found that the monochromatic blue image can be focussed satisfactorily on a transparent glass screen, by aid of a focussing glass, at a magnification of 1,000 diameters. It is then intended to calculate from the known difference of wave-length the change in focus required to bring the ultra-violet image into register. In order to reach the highest useful powers of these lenses, viz., 3,600 diameters, the image obtained on a fine-grained photographic plate at 1,000 diameters is to be enlarged in the ordinary way three and a half times.

Metallurgical Test Work.

There has been a very considerable increase in the amount of test work as compared with the past two years. It comprised two enquiries, jointly with the engineering department, into causes of failure of large steel objects; one investigation of steel rails for the Great Northern Railway; three type-metal enquiries; two of hardening of steel, and one of corrosion of brass. Photo-micrographic work was also carried out.

The features of these enquiries, which are of general interest, are as follows:

Failure of Large Steel Objects: Chemical analysis and microscopic examination were resorted to. In both cases the microscopic indications were sufficiently clear to allow of a definite conclusion being drawn. In one case failure was due to the steel being contaminated to an undue extent with enclosures of "slag" (probably silicate and sulphide of manganese). These are always present in commercial steel to some extent, but become a serious source of weakness when excessive. In this case the enclosures were never missing from a single field of view, even with magnifications of 150 diameters, when specimens several square inches in area were completely

searched. The material in question being subjected to extremely severe treatment, the microscope showed it to be fissured between adjacent enclosures. In the second case no serious defect in the structure of the mass of the steel was revealed, the only undesirable feature being the occurrence of some "ghosts" and other signs of segregation on a very small scale. Further examination of the material near the fracture proved that the metal had been subjected to very severe local deformation, and the nature and severity of this are regarded as sufficiently accounting for the failure.

The investigation of steel rails for the Great Northern Railway Company included a complete study of the microstructure, hardness and mechanical properties of various rails, some of which had been in prolonged use, and some were of novel composition or character. One rail which had been in use for 35 years proved to be low in carbon, but higher in phosphorus than most modern specifications would permit. The high proportion of phosphorus may have enabled the rail to withstand wear longer than a low-phosphorus rail of the same low carbon content and microstructure could have done, but this rail would probably not have lasted so well as the better modern rails under the present heavy main line traffic. The new rails varied so very widely in quality, as indicated by mechanical tests and microstructure, that they could not be classified. It will be interesting to follow the history of such of these types of rails as may be put into service, with a view of ascertaining how far the indications of tests may be relied on as predicting the service value of the rails. The method of "sulphur-printing" was used to test the distribution of sulphur in the cross-sections of the rails. It consisted in laying upon the smoothed surface of the rail section a sheet of bromide paper previously soaked for a short time in a 10 per cent. solution of sulphuric acid. Sulphuretted hydrogen is evolved wherever the acid in the paper comes into contact with a sulphide patch in the steel, and a corresponding patch of silver sulphide is formed on the paper, and thus the relative darkening of the various parts of the paper indicates the distribution of sulphur. The Great Northern Railway rails showed few signs of marked segregation.

Type Metal: In two cases the investigations included determinations of melting and freezing points, microstructure and chemical composition. The third enquiry related to a systematic investigation of type-metal in general with a view to improvement of manufacture. Unfortunately the laboratory was not able to continue the work under the conditions proposed by the firm. This is to be regretted, as the work already done gave much promise of valuable results.

Hardening of Steel: Specimens of steel hardened to various degrees by the aid of Messrs. Taylor, Taylor & Hobson's patent magnetic indicator for hardening steel were submitted to the laboratory, and after these had been reported on, that firm sent a small muffle furnace fitted with their patent indicator for trial. Subsequently they even presented the apparatus to the Laboratory, where it has proved useful for the occasional hardening of small tools. A person accustomed to the use of this indicator would, with normal varieties of tool steel (not high-speed or special steels), attain a very considerable degree of uniformity in hardening. In connection with his experiments on the distortion of steel during hardening, Colonel Crompton sent a number of enquiries to the Laboratory.

Corrosion of Brass: Some condenser pump rods which had broken after a relatively short period of use in seawater were examined. The microscope revealed that the brass had been vigorously attacked, the greater part of the zinc having been removed, leaving a mere spongy mass of more or less oxidized copper.

Market Prices During February and March.

During February the following prices prevailed:

Tin:—Opened at £125 per ton and rose sharply to £130 by the 4th, there were some variations until the 13th, followed by a fall

to £126 10 0 on the 18th, and a sharp rise to £131 on the 21st, the price then fell with some variations to £129 on the 29th.

English Lead:—Opened at about £15 4 0 per ton, and fell steadily during the month, reaching £14 2 0 on the 27th, the price rose slightly to £14 8 0 on the 29th.

Copper:—Opened at £61 per ton, rose to £62 on the 5th, and fell steadily to £56 15 0 on the 19th. It then rose to £58 8 0 on the 21st, and after slight variations reached about £57 10 0 on the 29th.

Hematite:—Opened at 58/10 and, with the exception of a rise to 59/- on the 5th and 7th, remained at this figure until the 19th when it rose to 59/2, remaining at this until the 25th it rose to 60/6 on the 28th, and fell from this slightly on the 29th.

Cleveland Pig:—Opened at 47/6 and rose with slight irregularities to 50/10 on the 26th, fell to 50/- on the 27th, and closed at 50/6.

Scotch Pig:—Opened at 56/-, rose to 56/6 on the 5th and 7th, and remained at 50/- until the 14th when it commenced to rise and reached 57/- by the 26th. It closed at 56/8.

The prices of chemicals at the end of February were as follows:

Ammonia sulphate, f.o.b. Liverpool.....	£12 2 6 per ton.
Copper sulphate	24 0 0 "
Caustic soda, white 77 per cent.....	11 2 6 "
Bleaching powder, 35 per cent.....	4 10 0 "
Carbolic acid liquid, 97/99 per cent.....	1 1 per gal.
Creosote, ordinary good liquid.....	2 1/4 "
Naphtha solvent, 90 per cent., at 160° C., f.o.b.	10 0 "
Shellac, standard T. N., orange spots.....	6 2 6 per cwt.
Antimony, Star Regulus, f.o.b.....	£34 to £35 per ton

The general tone of the metal market during March has been steady. Cleveland pig iron has risen throughout the month, closing at 52/- per ton. Copper showed an upward tendency during buying, but has since settled to £60. Electrolytic is quoted in sheets at £79 per ton. Block tin has risen to £142 10 0 to £143 10 0 per ton. Ingot and sheet lead shows slightly higher at £14 10 0 to £14 15 0 per ton. Zinc sheets are £25 10 0 per ton. Brass tubes (brazed) are at 9d per pound. Platinum is quoted at 100/- per ounce, and aluminium at £106 5 0 per ton, in ingots.

Hematite, after a sharp rise on the 3d of the month, remained fairly steady, finishing at 62/-.

Chemicals remain much at the same prices. Ammonia sulphate, f.o.b. Liverpool, £12 5 0 per ton.

Copper sulphate.....	£25 0 0
Caustic soda (48 per cent.).....	5 10 0
Bleaching powder (35 per cent.).....	4 10 0
Shellac, ordinary, per cwt.....	5 0 0
Para rubber, slight decline to.....	3 2 1/2 per lb.
Gutta percha.....	5/6 to 6/6

The Iron and Steel Market.

The pig iron markets have shown a very slight decline during April, while the prices of finished steel products have been held rigidly, in accordance with the policy formulated immediately after the decline in activity last October.

The total tonnage output in April was substantially the same as in March, but there was a slight decrease in new business, rather than an increase, which seems to presage slightly lower production in May and June. The general opinion expressed by steel interests, however, is that production will remain substantially unchanged until July, when the usual slackening will occur. Mills will close much more generally than usual for repairs in midsummer. Hopes are entertained that after the presidential nominations are made and crops assured there will be some improvement in orders.

No definite policy as to prices of finished steel products has been outlined in the public utterances of steel manufacturers other than that they should be just and reasonable at all times and that for the present no change is contemplated. Some

observers profess to be convinced that a general reduction will be made by the middle of June, the reductions being timed so as to give the market a chance to become settled before the usual fall buying period.

THE RATE OF PRODUCTION.

The rate of pig iron production increased in February over January, and in March over February, the March output being substantially the same as the December output. Since March there has been a slight declining tendency. The rate in the first four months of the year has been a trifle less than one-half the rate maintained last October, when the industry reached its zenith of productive activity, yet it has been slightly in excess of the rate during the phenomenal boom year of 1899, illustrating forcibly how productive capacity has increased in less than a decade.

The production of coke in the upper and lower Connellsville regions in the first four months of this year has been just a shade under 40 per cent of the production in the corresponding months last year, although the number of ovens has increased 12 or 13 per cent. The average furnace activity has been greater than Connellsville coke activity partly because furnaces having local ores and coke have done better than those depending on Connellsville coke and Lake Superior ores.

The output of finished steel has hardly been as great, relatively, as the output of pig iron, and has probably been at the rate, in the four months of this year, of about 40 per cent of the rate last October. The difference is accounted for partly by the accumulation of stocks of pig iron, some estimates, apparently based on good information, placing the pig iron stocks at more than a million tons.

At present the most active lines are tin plates and wire products; the moderately active are sheets and structural shapes, while the least active are pipe, rails, merchant bars and plates.

Tin plate production is substantially normal, and is likely to be maintained so until about the end of June, when it invariably declines. The leading interest is operating 202 tin mills; it has a total of 242 tin mills, but the bulk of the idle ones have not operated for two or three years. Almost all the independent tin mills are running full, and the current output is at least 95 per cent as great as ever reached. Tin plates normally constitute about 3.3 per cent of the total finished steel output.

The wire mills have been running at about 75 per cent normal, but stocks have been accumulated in jobbers' and retailers' hands, consumers not buying at the same rate the material has been produced, and somewhat lessened production is promised this month.

Sheet production has steadily increased and is now between 50 and 60 per cent of the maximum; current orders are at about the production rate, and neither an increase nor a decrease in output is promised for the next two months.

Structural business has been stimulated by the greatly reduced prices named by fabricating interests, whereby they seem to have given away about all their intermediate profit and have even been suspected, probably erroneously, of securing concessions on the mill shapes. The contract of the Corn Products Company, in the Chicago district, for instance, went at a flat price of \$44 a net ton, delivered, there being 3,600 tons of fabricated work and 600 tons of castings. Prices of fabricated material have declined \$10 to \$15 a ton from last year, while there has been no open change in structural shapes since the advance of August 31, 1905. Most of the structural mills are running to part capacity, and current output is probably more than 50 per cent of normal.

Business in steel pipe is diminishing. Present prices have been guaranteed to May 15 and jobbers, now that they have had an opportunity to dispose of their stocks, are anxious to see a reduction, as the present price basis is seven points, or about \$14 a net ton, above the basis ruling at the beginning of 1906. Other steel products have been advanced relatively little during that period.

The rail trade is unsatisfactory, partly on account of the poor

financial condition of the railroads and partly on account of the difficulty over specifications. Based on the track mileage of the roads which have formulated their requirements for the year it has been estimated that standard rail business may not be more than 30 per cent the average of the past two years, but the roads may, of course, come in with additional requirements later.

The Pennsylvania Railroad Company has made public the specifications which accompanied the orders for 55,000 tons which were formally tendered the rail mills on February 6 and refused. The most objectionable features are that if a rail breaks under test without showing physical defect the entire heat is summarily rejected; that if it does not break, but on testing to destruction shows pipe, the top rails of the heat shall be rejected, and that "no rails shall be accepted which contain any physical defects that impair their strength," while there are more stringent requirements as to the amount of straightening permissible in the cold straightening presses. It is not permitted to hold the piece, or use artificial means of cooling, from the leading pass to the hot saws, and the shrinkage allowance is a trifle less than that in former practice, requiring finishing at a slightly lower temperature. The matter of rail specifications seems now to have developed into a contest of endurance between the railroads and the rail mills, the railroads having this advantage that they would not be disposed to buy rails freely in any event.

Merchant bars are inactive, and are suffering from the competition of bars rolled from old steel rails. A price as low as \$1.75, delivered, has been made on reinforcing bars for concrete, made from old rails, while steel bars are \$1.60, base, Pittsburgh. The season cotton tie price was announced in April, at 85 cents a bundle, 10 cents less than the 1907 price, and the same as ruled in 1905 and 1906. Plates are probably the dullest item in the list, as the steel car business is all finished and there is no prospect of further buying for the present.

Iron bars are being shaded more than formerly, particularly in the East. The recognized market is \$1.50, Pittsburgh, for Pittsburgh or Eastern delivery, and \$1.47, Pittsburgh, for Western delivery. Steel prices are being shaded occasionally in some lines. Delivered prices on billets and sheet bars have been revised, but the base, Pittsburgh, remains unaltered. With a few exceptions delivered prices are the base prices of \$28 on billets and \$29 on sheet bars, plus one-half the freight from Pittsburgh to point of delivery, but for actual Pittsburgh delivery billets are \$28 and sheet bars \$29.50. The regular official prices on finished steel products remain as follows:

Beam and channel, 15-inch and under, \$1.70.
Plates, tank quality, \$1.70.
Steel bars, \$1.60, base, half extras.
Wire nails, \$2.05, base.
Sheets, 28 gauge, \$2.50.
Tin plates, 100-lb. cokes, \$3.70.

The Outcome of the Bearing Metal Alloy Suit.

The Ajax Metal Co., of Philadelphia, in their suit against the Brady Brass Co., of Jersey City, N. J., had formerly obtained a decision from the Circuit Court of the United States for the District of New Jersey sustaining the validity of patent 655,402 (owned by the Ajax Metal Co.) and finding infringement by the Brady Brass Co.

This decree of the lower court has now been reversed by the United States Circuit Court of Appeals for the Third Circuit, and patent 655,402 of the Ajax Metal Co. has been found invalid. The decision was rendered by Judge Gray.

The patent in question is a product patent for a bearing metal containing "less than 7 per cent tin and more than 20 per cent lead and the balance copper." Before this patent was applied for, it was known in the trade that the quality of the bearing metal would be improved by increasing the amount of lead and that it was necessary for this purpose to reduce

the tin. The selection of less than 7 per cent tin and more than 20 per cent lead is considered a mere change in proportions toward which the state of the art was already drifting, and which therefore did not involve an act of invention or introduce any changes other than those already in progress in the normal development of the art. Satisfactory evidence was also produced that such an alloy had successfully been made by the Brady Brass Co. The decision concludes that "a mere advance in the proportions of the constituents of an alloy, however useful the result may be, does not entitle the originator to the monopoly of a patent, in the absence of other circumstances, than those here disclosed."

Exhibition of Safety Devices.

The exhibition at the Museum of Safety Devices, 231 West Thirty-ninth Street, New York, was formally opened on April 11. Speeches were made by Mr. Walter C. Kerr, Prof. F. R. Hutton, Dr. W. H. Tolman, Dr. Josiah Strong, Prof. C. E. Lucke and Mr. Charles Kirchhoff.

While many of the exhibits relate to the prevention of railway accidents, the exhibition is of general and diversified interest. The United States Steel Corporation is planning an exhibit of steel props for coal mines. They will be shown in a full-sized model of a coal shaft. The appearance of the interior will be simulated in papier-mâché, and a miner will be at work to make the exhibit more realistic.

CORRESPONDENCE.

Transvaal Rock Drill Competition.

To the Editor of *Electrochemical and Metallurgical Industry*:

SIR:—As you will see from the following notice, the Transvaal Government and my Chamber are initiating a stope drill competition which will be held in the early part of next year. It is desired to give this fact the widest publicity in order that drill manufacturers in America, as well as in other countries, may be afforded the opportunity of taking part in the competition.

TRANSVAAL CHAMBER OF MINES (Johannesburg),
LONDON, March, 1908.

A. N. GOLDRING, *London Secretary*.

TRANSVAAL STOPE DRILL COMPETITION.

The Transvaal Government, in co-operation with the Transvaal Chamber of Mines, has arranged for a practical trial of small rock drills suitable for narrow stoping work under the working conditions obtaining on the Witwatersrand.

All types of rock drill are eligible to compete. Drills using compressed air will be supplied with a pressure varying from 60 to 75 lb. per square inch at the working face. The mining regulations require the provision of dust-allaying appliances, and competitors must make provision accordingly.

Two prizes, of £4,000 and £1,000, respectively, are offered. The trials and the judging will be so arranged as to decide which machine performs the most economical work.

The competition will commence early in 1909, and entries will probably close with the end of 1908. The trials will last about six months, the drills being tested in the first instance on the surface and those considered suitable being given a more prolonged test under ground in several stopes in various mines on the Witwatersrand.

The detailed conditions governing the competition, including the exact date of closing entries, will be published as soon as possible.

All enquiries should be addressed either to the secretary, Stope Drill Competition, Transvaal Chamber of Mines, Johannesburg, Transvaal, or to the London secretary, Transvaal Chamber of Mines, 202, 203 and 206 Salisbury House, Finsbury Circus, London, E. C., England.

The Power Plant of the New Addition of the Raritan Copper Works.

By FRANK D. EASTERBROOKS.

With the increased production of copper at the mines and smelters came the necessity for additional refining capacity. The Raritan Copper Works (for a description of the original plant see the article by Lawrence Addicks, *Mineral Industry*, Vol. IX, 1901, page 261), designed and built by its manager and superintendent, Mr. J. C. McCoy, had a maximum monthly tank house production of 13,000,000 to 14,000,000 pounds of cathode copper. The new plant, representing in its construction the best modern engineering practice applied to electrolytic

and weighed, it is unloaded in the yard storage with a Gantry crane, thus releasing these cars for further use.

A new coal trestle of 5,000 tons capacity has been constructed close to and parallel with the power house. That portion opposite the boiler room is used for storing anthracite coal for the boilers, while on the rest of the trestle bituminous coal, coke, pyrites, etc., for the copper furnaces are stored. Electric cars running on tracks under the concrete floor of the storage are used to transfer the material to the furnaces.

For fire protection all of the buildings, with the exception of the furnace buildings and No. 2 power house, are equipped with an automatic sprinkler system. Two separate brick buildings house the 1,500-gallon and two 1,000-gallon per minute underwriter pumps, which are always ready for immediate use, while

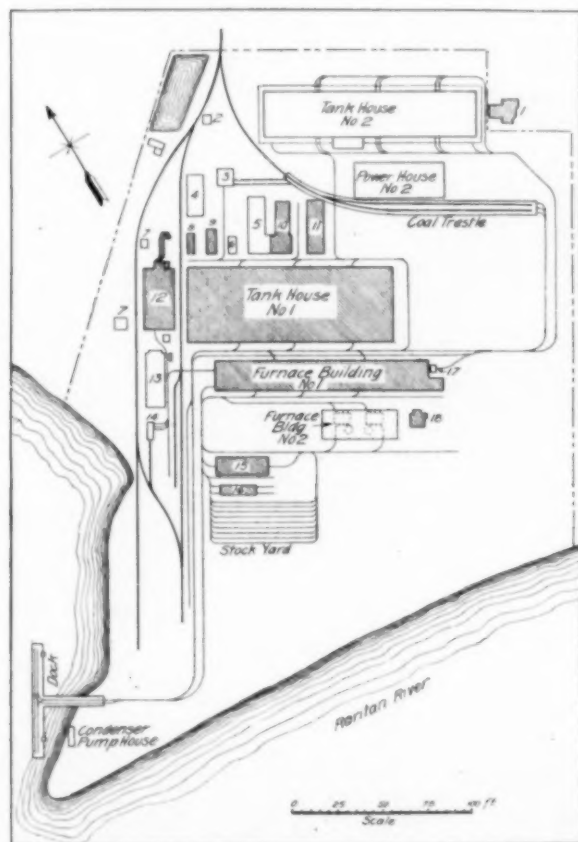


FIG. 1.—PLAN OF RARITAN COPPER WORKS.

- | | |
|-------------------|-----------------------|
| 1. Office | 10. Cupel house |
| 2. Oil Storage | 11. Sulphate building |
| 3. Foundry | 12. Power house No. 1 |
| 4. Machine Shop | 13. Coal storage |
| 5. Parting plant | 14. Engine house |
| 6. Smithy | 15. Storage building |
| 7. Fire pumps | 16. Sampling |
| 8. Carpenter shop | 17. Elevator |
| 9. Stores | 18. Blast Furnace |

refining, was designed and built by Mr. George K. Fischer, consulting engineer of the company, under the immediate direction of Mr. D. J. Nevill, member Amer. Soc. Mechanical Engineers, and has a maximum output approximately the same as the old plant.

On the accompanying plan (Fig. 1) is shown the general arrangement of the entire works, the shaded portion representing the old part.

The system of narrow-gauge tracks has been extended throughout the entire plant, three Porter locomotives and a large number of 10-ton flat cars being used to transfer the material. A standard-gauge switching engine is part of the equipment. Incoming copper is unloaded from freight cars by air hoists onto the small steel-frame cars of the narrow gauge system. If not required for immediate use, after being sampled

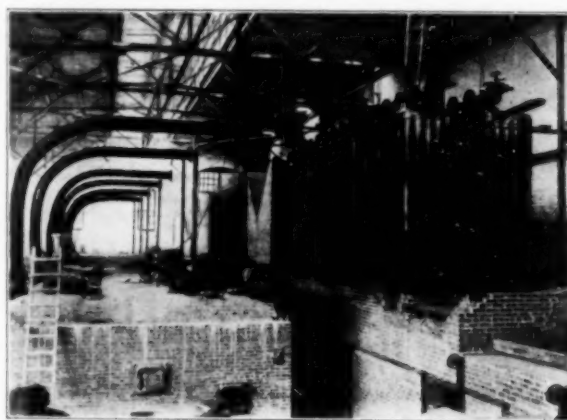


FIG. 2.—GENERAL VIEW ON TOP OF BOILERS, SHOWING FLUES AND TWO ECONOMIZERS.

a 100-gallon pump is used to maintain a constant pressure on the lines. A 500,000-gallon reservoir and a number of hose houses situated at convenient points form a part of the protective system, which includes also a finely organized fire department.

Power House, No. 2.—This building, in which are located the electrolytic generating units for the No. 2 tank house,



FIG. 3.—ANOTHER VIEW OF ENGINE ROOM.

(Note the massive foundations.)

the air compressor, and all the light and power generating units for the entire plant, is 260 ft. long by 106 ft. wide. It is a steel-frame structure with concrete foundation and brick walls. On top of the fireproof roof, which was constructed by the Metropolitan Fire Proofing Company, is a five-ply tar and slag roof. A brick wall running lengthwise of the building separates the engine room from the boiler room. The rein-

forced concrete floors in the engine room were designed to carry a load of 200 lb. per sq. foot on the panels. The engine room contains a 30-ton three-motor Whiting crane of 50-ft. span by means of which the armatures of the four electrolytic generators were placed without difficulty.

In the construction of the power plant every precaution was taken to guard against possible interruptions of service, which

provided with individual damper control, while the main flue damper is operated by a Thompson damper regulator. The side walls of the economizer chambers are made of removable asbestos sections, which construction has a decided advantage over brick walls in the event of replacing tubes. The soot scrapers are driven by a belt-connected motor.

Stack.—The self-supporting steel stack is 200 ft. high above

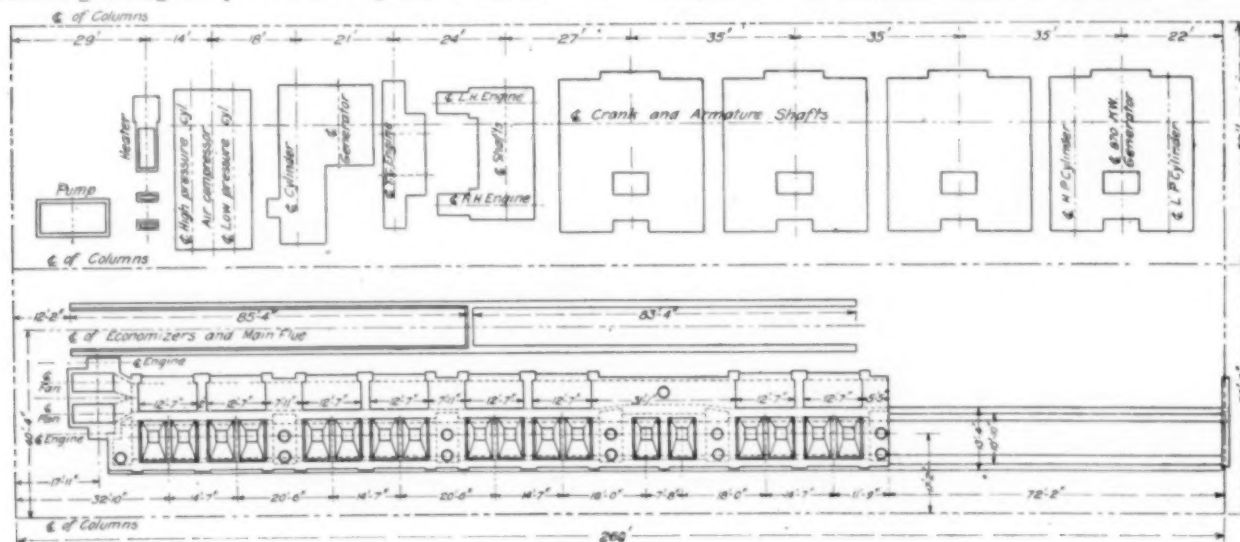


FIG. 4.—FOUNDATION PLAN OF POWER HOUSE.

would interfere with tank-house operations and reduce the output of cathode copper. The power cost is about 40 per cent of the total cost of refining copper from anode to cathode, but the steady 24-hour load on the electrolytic generators gives ideal conditions for the production of power cheaply.

Boiler Room.—The boiler plant consists of eight Babcock & Wilcox water-tube boilers, set in batteries of two each, with a heating surface of approximately 4000 sq. ft., comprised in three 36-in. drums, 29 ft. 4 in. long, and 189 4-in. tubes 18 ft. long, divided into 21 sections of nine tubes each. A steam pressure of 160 lb. per square inch is carried.

The drums and sections of the boilers are hung from a steel frame, while the brickwork was built up independent of the frame and is unaffected by the expansion and contraction. The bricks were laid in a lime-cement mortar and tied in with a full header course every fourth row, to give maximum strength and prevent development of cracks. Each boiler is provided with three 5-in. pop safety valves.

The furnaces are fitted with McClave grates of 112.5 sq. ft. area. They are 9 ft. deep by 12 ft. 6 in. wide and are hand-fired. The buckwheat anthracite coal is dumped directly in front of the boilers from hopper bottom cars run up on to the coal trestle. An ash tunnel runs the length of the boiler room under the furnaces and connects with the tunnel under the coal trestles. The ashes drop into brick-lined steel hoppers fitted with gates, and are removed in side-dumping steel cars. Under-grate blast is furnished to the furnaces through a concrete tunnel back of and parallel with the ash tunnel by two 84-in. Sturtevant fans direct-connected to 7 in. by 10.5 in. engines, which are under automatic control with the main flue damper.

Flues and Economizers.—The general arrangement of flues and economizers is shown in Fig. 6. The flue gases, after leaving the boiler, pass through the economizers, which are located directly behind the boilers over the flues, into the stack. One economizer of 48 sections of 12 tubes each containing approximately 15,700 sq. ft. of heating surface, is provided for each four boilers. The arrangement of dampers is such that the gases can be by-passed directly to the stack. Each boiler is

provided with individual damper control, while the main flue damper is operated by a Thompson damper regulator. The side walls of the economizer chambers are made of removable asbestos sections, which construction has a decided advantage over brick walls in the event of replacing tubes. The soot scrapers are driven by a belt-connected motor.



FIG. 5.—INDIVIDUAL SWITCHBOARD FOR ELECTROLYTIC CIRCUITS. (Capacity, 10,000 amperes. The square box on the right is Eppensteiner relay.)

tion by fifteen 3-in. bolts. The brick lining is supported every 20 ft. by 5-in. channels riveted around the interior of the stack. A ladder runs from the base to the top of the stack, and on the top is a painting ring. Three coats of Toch Bros. stack paint were applied after erection, the color of the second coat being changed to permit of better inspection.

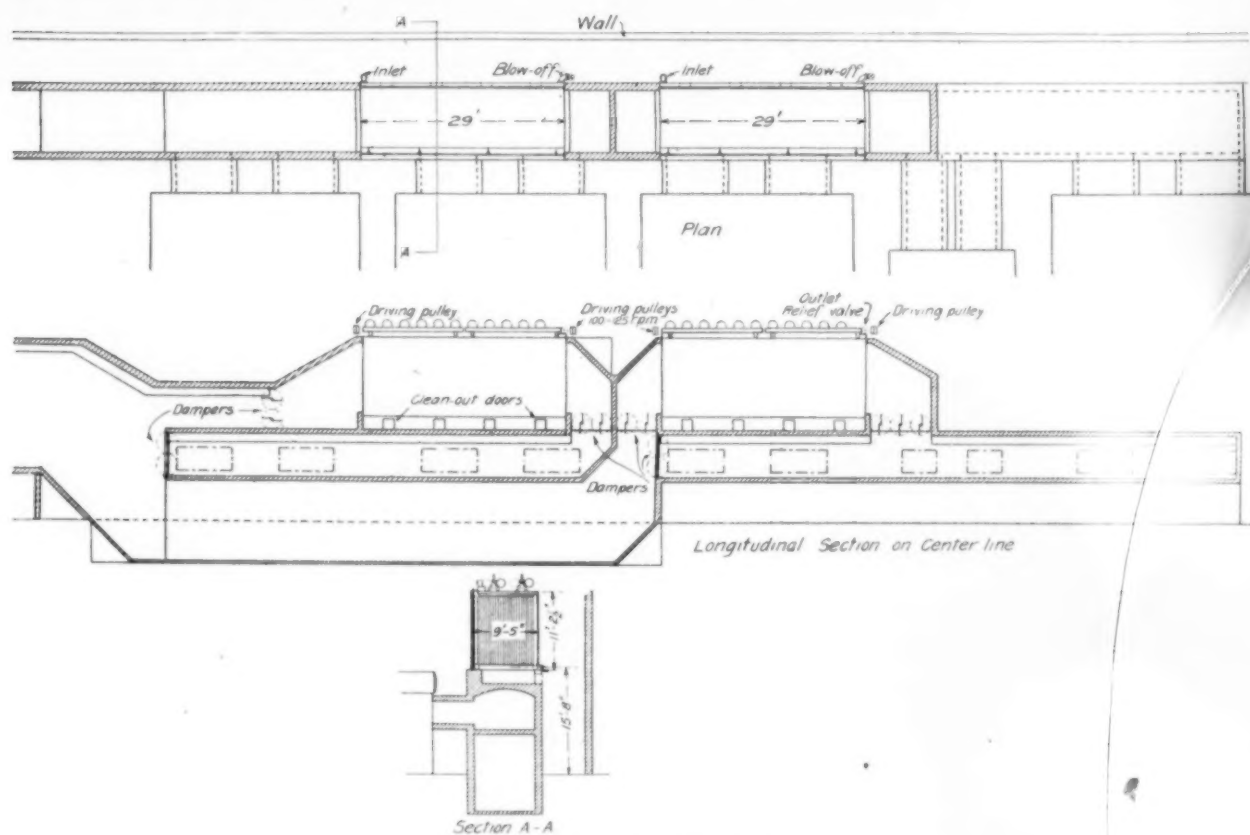


FIG. 6.—FLUES AND ECONOMIZERS.

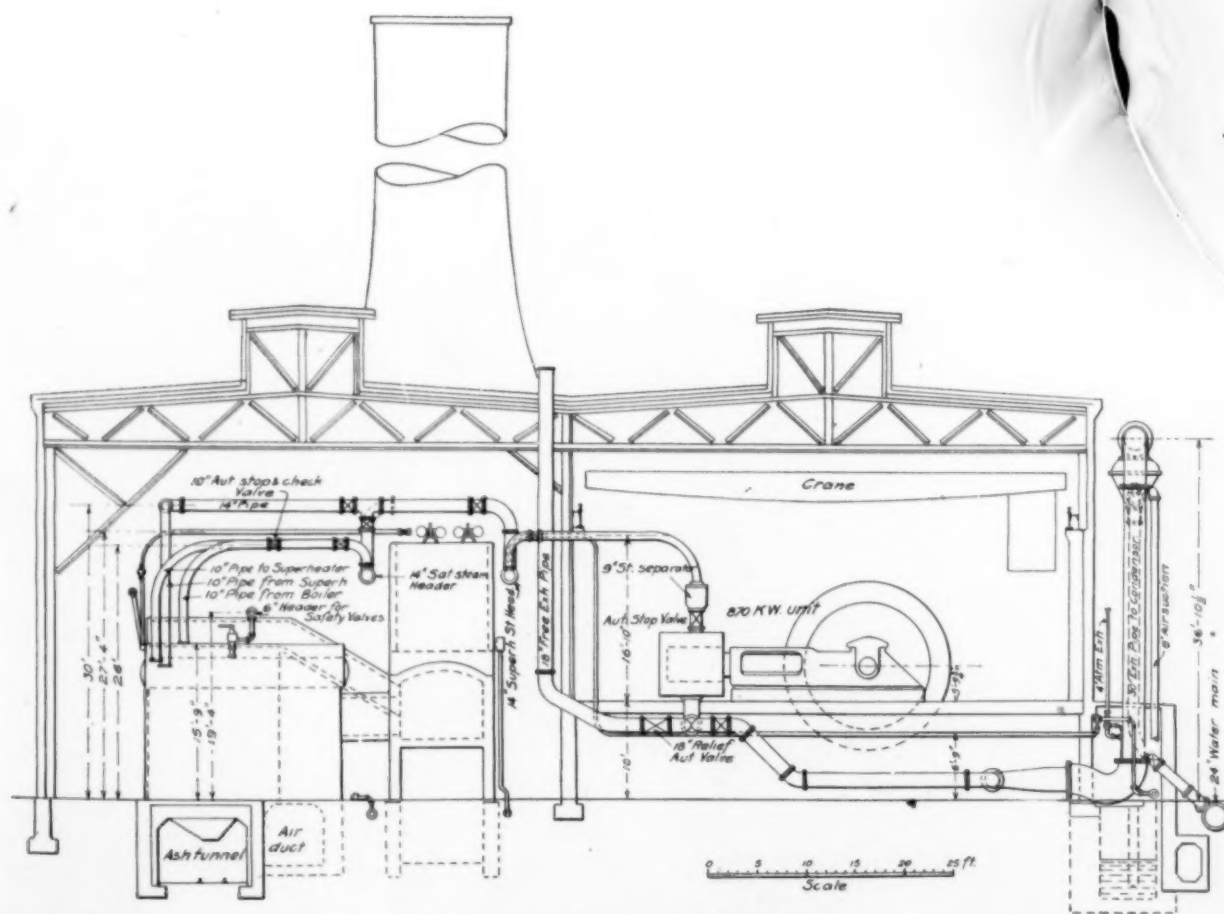


FIG. 7.—SECTION THROUGH POWER HOUSE.

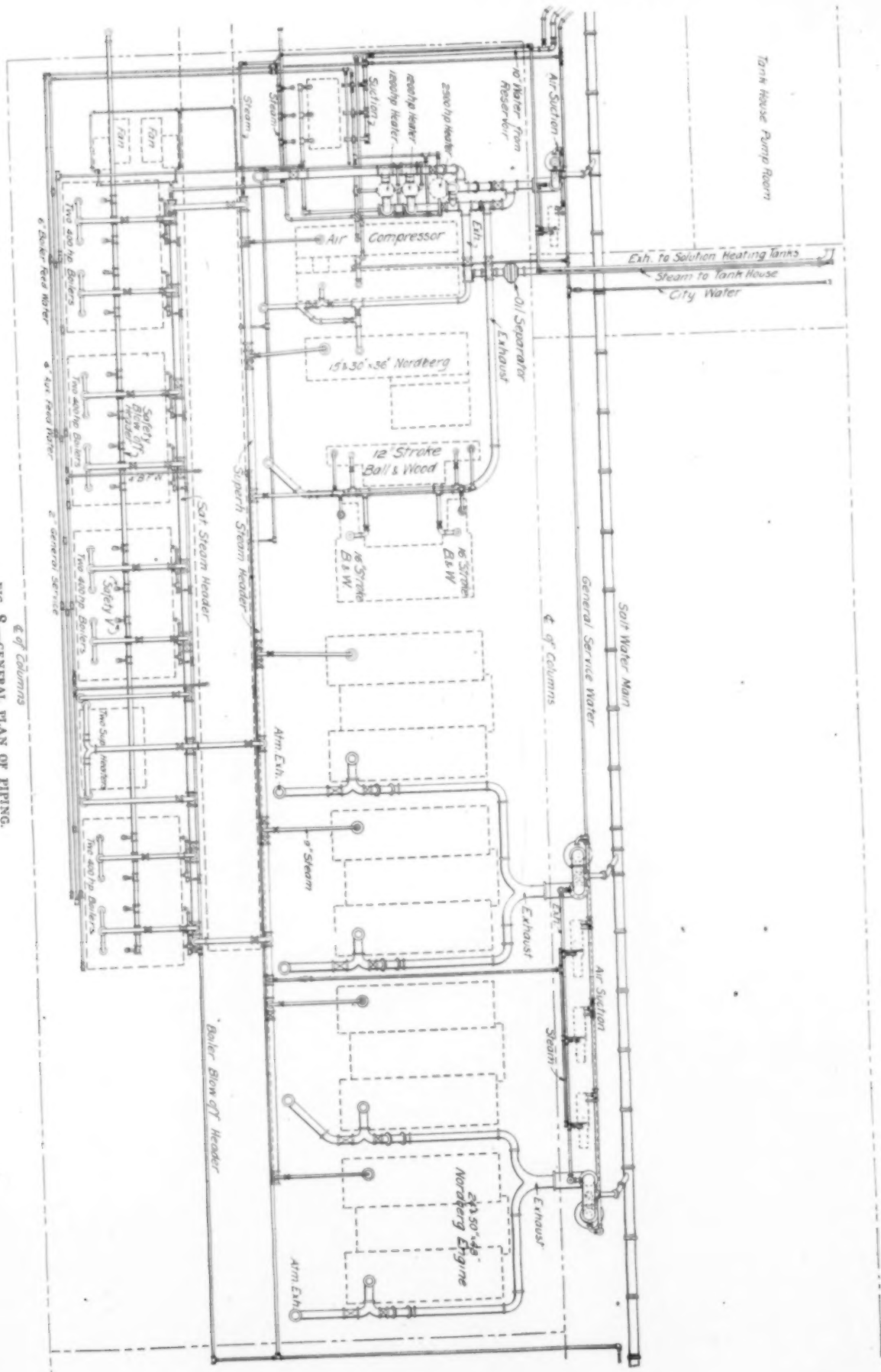


FIG. 8.—GENERAL PLAN OF PIPING.

Engine Room.—For furnishing power to the three electrolytic circuits in the tank house there are four 24 in. x 50 in. x 48 in. horizontal cross-compound Nordberg engines especially designed for heavy duty, driving 870-kw General Electric generators. One unit, held in reserve, is connected to run on any of the tank-house circuits. These engines have Corliss valve gear, run at 100 r.p.m. and are guaranteed an economy at full load of 11.4 lb. of steam per i.h.p. with 150 lb. steam pressure at the throttle, 100 deg. F. superheat, and 28 in. vacuum. They are provided with Locke engine stops. The individual switchboards and switches were constructed by the Walker Electric Company. Two switches in parallel have a capacity for the full overload of 7500 to 8000 amperes. Eppensteiner engine stop relays, manufactured by the Weston Electric Company, are in use on the switchboards. This relay, which was worked out at the plant of the U. S. Metals Refining Company (see article by Addicks, *Eng. and Min. Jour.*, May 25, 1907), is a valuable protective device for electrolytic circuits. It is connected to the ammeter shunt and in the event of an overload operates the engine stop and shuts down the engine.

For furnishing light and power to the entire plant there are four Ball and Wood horizontal, tandem-compound, high-speed units, two direct-connected to 75-kw generators and two driving 175-kw General Electric generators, which were transferred from No. 1 Power House, and a 15-in. x 30-in. x 36-in. horizontal cross-compound Nordberg engine running at 110 r.p.m. and driving a 275-kw General Electric generator. These units all operate in parallel at 240 volts direct current. The distribution of power to the various departments is controlled from a main switchboard, which was furnished by the Walker Electric Company, and is equipped with Weston instruments and I. T. E. circuit-breakers of the Cutter Company.

A Nordberg horizontal, cross-compound two-stage air compressor 13-in. x 20-in. x 36-in. stroke at the steam end and 14.25-in. x 23-in. x 36-in. at the air end, with a capacity of 1500 cu. ft. of free air per minute furnishes air for the numerous uses around the plant. This compressor has an economy of 11.5 lb. of steam per indicated hp running at 87.5 r.p.m.

Piping.—The general plan of the piping is shown in Fig. 8. Normally feed-water is taken from artesian wells sunk on the premises, the water being pumped into the reservoir by compressed air. Three Blake compound outside-packed plunger pumps 8-in. x 7-in. x 12-in. force the water through the closed heaters, and the economizers into the boilers. An auxiliary feed line takes water from the city water company. The three closed heaters, made by the Goubert Manufacturing Company, are connected so that any or all can be readily by-passed if found necessary for any cause. A Venturi water meter, with self-registering device, measures the water feed to the boilers. A separate supply line running to the boilers furnishes water for washing down, testing, etc.

Long-radius pipe bends and extra heavy welded steel flanges were used throughout. All of the valves used on the main steam lines are of the gate type, furnished by Best & Company, with the exception of the automatic stop and check valves installed on the boilers, which were furnished by the New Bedford Boiler & Machine Company. The steam lines are suspended simply by turn-buckle rods, hung from trussed beams, resting on the roof trusses of the building. Two Foster superheaters, separately fired and located in a line with the boilers, are used to superheat the steam. Each one is capable of superheating 48,000 lb. of steam per hour 125 deg. F. The duplicate steam mains with their cross-connections enable any of the engines to be run on either saturated or superheated steam. Provision is made through a proper arrangement of pipes and valves for frequent economy tests of both boilers and engines.

The exhausts from the engines running on the electrolytic circuits pass into two Worthington elevated jet condensers, one 30-in. condenser being connected to the exhausts from two engines. Three dry-air pumps, each 10-in. x 18-in. x 18-in.,

are connected to these two condensers. Condensing water is supplied to both power houses by three 15-in. R. D. Wood centrifugal pumps direct-connected to motors which are located near the river in a separate building.

The light and power engines and the air compressor are arranged to be run condensing or on back pressure, the exhaust steam being used to heat the tank-house electrolyte. The exhausts connect to an 18-in. jet condenser, to which is connected a 6-in. x 12-in. x 12-in. dry-air pump.

The usual gravity return system is used for circulating the lubricating oil.

The following table gives some of the principal features of the two power houses:

	No. 1 Power House.	No. 2 Power House.
Boilers.....	Aultman & Taylor,	Babcock & Wilcox,
Grates.....	Murphy stoker,	McClave hand-fired,
Coal used.....	Bituminous,	Anthracite,
Steam.....	Saturated,	Superheated,
Engines.....	Ball & Wood vertical,	Nordberg horizontal,
Generators.....	Westinghouse and General Electric,	General Electric,
Maximum current	5000 amperes.	7500 to 8000 amperes.

ORE DRESSING

With Special Reference to Oil Concentration

An interesting review of the present status of the art of ore dressing was given by Mr. W. G. SWART in his recent presidential address before the Western Association of Technical Chemists and Metallurgists. The address is printed in full in the March issue of the *Western Chemist and Metallurgist*.

In mechanical separation the operator is brought in every instance early and squarely against the slime or dust problem and "to-day there is no successful method of handling either for purposes of separation." But along the line of separating particles of medium fineness considerable progress has been made.

As to wet concentration Mr. Swart suggested three different lines of work as promising.

First, the use of hot water, which is said to have already attained some considerable degree of success.

Second, the use of alkaline or saponaceous material in the mill water to aid in the rapid settlement of slimes, in the clearing of wash water where it must be used over and over, and in the prevention of certain surface-tension effects.

Third, the application of centrifugal force to increase artificially the relative difference of density between minerals usually classed together.

As to dry concentration of ore some facts have now been more clearly established than ever. First, dust cannot be concentrated and must be removed. Second, sizing must be very much closer than for wet concentration, which brings up the screening problem. Third, the active body of air must be divided up into small sections, since it is not practically possible to handle ores in large masses continuously, as for example on the Hartz jig. Finally, the dry mill has a labor problem of its own, since the ordinary millman does not understand clearly the underlying phenomena.

Mr. Swart says that the best and most successful attempt at dry concentration yet brought to his attention is the work done by Messrs. Sutton, Steele & Steele, of Dallas, Texas.¹ "If dry concentration—or more accurately, pneumatic concentration—can succeed, it would seem to be along the lines of their dry concentrating table. There are no new principles involved—simply a skilful and intelligent mechanical application of old ones, of which the limits have long been known. Much the same thing might be said of the original Wilfley table, yet its combination of old principles was novel and simple, and its

¹See the descriptive article on another page of this issue.

use has revolutionized the methods of ore dressing in many districts. A successful dry process, including necessarily the preliminary preparation of the ore with the concentration devices proper, would undoubtedly work a similar revolution."

The difficulties met with in pneumatic concentration also exist in magnetic and electrostatic processes, although not to the same degree. The dust problem is always serious, yet, due to the fact that most of the successful work in this direction is done on a concentrate or middlings product from preliminary wet concentration, most of the dust having been eliminated as slime, these processes have been able to more than hold their own. Their chief field seems to be in the separation of minerals whose closeness of density does not allow of a separation being made wet or dry through difference of specific gravity.

Magnetic separation is confined for all practical purposes to mixtures containing iron in some form. It has been developed along two lines; first, the use of low-power magnets on ores having a highly permeable iron product for removal; and second, the use of an intense or concentrated magnetic field on materials which are of low permeability. With the exception of magnetite, nearly all ores to be treated by the low-power machines must be roasted. Most of such commercial work now being done is in the removal of pyrite or marcasite from zinc blende.

A comparatively large tonnage of ore is being daily produced by this method, which is commercially and metallurgically successful, and undoubtedly one of the most reliable. It has the advantages that little skilled labor is required, the roast is ordinarily not expensive nor difficult, and the separation can be made as coarse as the crystallization will allow. The machines have large capacity and are simple. The zinc product is high grade and clean, if the preliminary work be well done.

The disadvantages are the large losses in roasting and in tailings, running as high as 35 per cent of the zinc contents in some Wisconsin mills, but as low as 15 per cent in some Colorado practice on finer material. In addition to these must be considered the loss of the sulphur value of the pyrite, the damage in some cases from the sulphurous gases, the difficulty in satisfactorily re-roasting a blende once partially roasted, and the fact that much marmatite possesses about the same magnetic permeability as indifferently or partially roasted pyrite, being therefore lifted out and lost with it. Any galena present is also left with the blende.

A very considerable tonnage of Western marmatite ore is being successfully handled without roasting by the Wetherill magnetic separator and others using an intense magnetic field. These machines lift out the marmatite by reason of the combined iron, leaving the pyrite and galena behind. The avoiding of the roast, and the elimination of the galena from the blende, give this method a most decided advantage over roasting, but like most other processes it is limited in its scope to suitable ores, and there are many ores which it will not handle at all, or only if used in combination with some other process.

This is partly due to the fact that pyrrhotite and sometimes pyrite after heating (which is necessary for drying) possess a magnetic permeability overlapping that of the accompanying blende, and hence are lifted off with it, contaminating it. There is also the consideration that marmatite is variable in composition, the percentage of combined iron running from 3 per cent to 30 per cent. Any "rosin blende" or marmatite low in iron fails to respond to the magnets and is therefore left behind to contaminate the pyrite and galena.

Magnetic separators, particularly those of the belt type (which are in most common use), are further troubled with entanglement of the particles to a greater or less degree. The vibration of the machine, together with the movement of the belt, tends to sort or size the particles, the fines working down through the coarse and lying next to the belt surface so that they are hindered in rising again through the coarser particles

to the magnets. This difficulty increases as the capacity of the machine is raised.

However, the difficulties are not greater than in other branches of ore dressing; they are only different in kind.

Mr. Swart then passed over to a discussion of electrostatic separation and its difficulties.

"The electric charge is like water or air under pressure; leaks are fatal to efficiency." It has been determined that the chief source of static leakage is not through the air or gaseous medium in which the generating plates revolve, but over the supporting, insulating and generating surfaces, due to dirt, chemical films or condensation of moisture. The two former can be guarded against by furnishing the machine with a copious supply of fresh air from which dust and dirt have been filtered, and chemical action reduced by this same rapid change of air. Condensation is prevented by the simple expedient of heating this air supply, the raising of its temperature only a few degrees being sufficient. This remedy is simple and effective and under such conditions a good static machine has been run thirty days without cleaning. The efficiency gradually drops, however, and toward the end of the campaign the electrical output is comparatively small. This varying charge necessitates corresponding adjustments on the separators.

Some fairly satisfactory results have been obtained by running the static machines under a pressure of several atmospheres, but the mechanical troubles offset any gain in efficiency.

"But the goal has been not an improved static machine, but the utilization of the dynamic current, and I am pleased to be able to state that this has been accomplished in Boston by a company of clever and intelligent workers along this line of electric separation. The apparatus consists essentially of a generator, transformer, rectifier, interrupter and condensers—all well understood and thoroughly tried pieces of apparatus. It seems beyond question that this method has already superseded the static machine with a corresponding certainty of results and accuracy of regulation."

Mr. Swart finally takes up "a series of methods of comparatively recent use, but of most intense interest. These are the flotation processes, the oil processes and the surface-tension process. I group them together because I have some reason to believe from results obtained by careful experimenting, and by comparison with the results of others, that the fundamental principles underlying these processes are electrical and identical. I mean that in all probability the law underlying the attachment of a gas bubble, or a film of oil, or a film of air, to a particle of zinc blende in any one of these processes is the same, and has to do with a property or group of properties pertaining to the blende and the medium, varying perhaps in degree, but ultimately the same."

In working out the details of these several processes, the finest sort of practical work has been done. The knowledge gained of the selective action of various oil and grease mixtures, of the use of acids, of the use of varying pressures—running from a high vacuum to several atmospheres—of the effect of temperature, time and of the amount and character of agitation, has added enormously and directly to their commercial value.

"It seems practically a certainty that the flotation processes will eventually be made perfectly selective, separating one sulphide from another as well as sulphides from quartz or other gangue materials, through the use of proper solutions, regulated as to density and temperature, and with regard to the time of treatment and other variable factors. Just what can be done by artificial means toward varying the surface tension of water so that selective action may be obtained in what is known as the surface-tension process, remains to be seen."

Mr. Swart concludes by emphatically urging the necessity for accurate knowledge of theory. We are collecting much data, but not systematically. Most of our work is done in the dark. We are hampered by purely commercial considerations.

"We ought also to begin to take home to ourselves, as definitely bearing on our metallurgical work, the recent researches

and discoveries as to the ultimate constitution of matter—the intra-atomic theory. I feel sure the way to success lies there.”

* * * * *

Some notes on the use of various recent concentration methods for winning zinc concentrates from Broken Hill tailings are given by C. GOEPNER in *Metallurgie* of February 22. In the half year ending May 31, 1907, the Broken Hill Proprietary Co., Ltd., treated 116,240 tons tailings by the Delprat flotation process and produced 25,353 tons of zinc concentrates. The proportion of zinc was 41.81 per cent and increased up to about 43 per cent during the last weeks of that period. It is believed that this improvement was due to the use of deeper pans for separating the concentrates from the sand. It has been found that in order to be successful the material to be treated by the Delprat process must be sufficiently comminuted and for this reason a plant for regrinding the coarser tailings has been installed.

It is said that in the treatment of rich tailings the Delprat process has so far given good financial results. If the output is not of decisive importance, much may be accomplished with it, but it is not an ideal process, and it is possible that it may have to be replaced by other processes in future.

The report of the Zinc Corporation, Ltd., is less favorable. This company had intended to use the Potter flotation process. After preliminary tests made by A. L. Queneau from July to October, 1906, it was decided to introduce the Potter process on a large scale, to be followed by magnetic concentration, in order to enrich the zinc concentrates and obtain a lead concentrate as by-product. The process was used on a large scale from the end of January to the middle of March, 1907. Three thousand one hundred and eighty-three tons crude tailings were treated and there were obtained 493.3 tons of zinc concentrates with a content of 40.1 per cent zinc, 9.21 per cent lead and 9.78 ounces silver per ton; besides this 32.39 tons lead concentrates containing 49.9 per cent lead and 21.4 ounces silver per ton.

The efficiency of zinc recovery from the tailings was 46.28 per cent. But while it had been expected that 500 tons of tailings could be treated per day it was possible to treat only 150 tons. The value of the concentrates was \$2.07 per ton of tailings. If the magnetic concentration had accomplished what had been expected this value could have been increased to \$3.04, but the magnetic concentration plant was not yet ready and could not be used. But it had become evident that on account of very low output and high cost the Potter flotation process was commercially impossible.

Under the management of Huntley an experimental plant was then erected using the Cattermole process, which employs acids and oil. After favorable results had been obtained in the laboratory, the process was carried out in one apparatus on a larger scale. Three thousand, nine hundred and seventy tons of crude tailings were treated and 1091 tons of concentrates were obtained, containing 44.9 per cent zinc, 9.6 per cent lead and 10.1 ounces of silver per ton. Seventy-three per cent of the zinc contents was recovered. The value of the concentrates per ton of tailings treated was \$4.50.

While it was found that the process was satisfactory with respect to output and concentrates, provided the operation was carried out carefully, this was not thought to be sufficient. In order to reduce the cost to a minimum, it was necessary to push the output to the utmost, and in this respect difficulties were encountered. It was then decided not to use the Cattermole process on a large scale, but to introduce the Elmore process instead. It is said that the two processes are identical in many respects and only in the last step of the mechanical separation of the concentrates there are real differences between the two processes. But the Elmore process is said not to need such comminution of the particles and to operate at ordinary temperature, nor does it require skilled labor, while very reliable men are required for operating the Cattermole process.

Together with the Elmore people the Zinc Corporation in-

stalled first an apparatus treating 30 tons of concentrates per day. Rich tailings from Block 10 mine were treated and 89 per cent of the zinc were extracted. Three further experiments with tailings from the British mine gave 83 to 89 per cent extraction and concentrates of 44 to 45 per cent zinc. These results are better than had been expected. The intention is to install first 16 Elmore apparatus and if the results are good to double the installation.

* * * * *

In this connection it will be interesting to give a brief sketch of the Elmore oil process on the basis of some recent publications (B. HAYWOOD in *Western Chemist and Metallurgist*, March; C. GOEPNER in *Metallurgie*, Jan. 8 and 22; R. LINDE in *Metallurgie*, Feb. 8).

Like the processes of Potter and Delprat,² the Elmore process is a flotation process. But while the processes of Potter and Delprat are acid processes, Elmore uses oil to float the metallic particles. It is, however, not the oil alone, which on account of its low specific gravity raises the metallic particles. This became clearly evident from two observations made at an early date.

The first is that the specific gravity of the oil itself is not of such decisive importance as might be thought. The second is that in practice it was found that the oil was able to float two or three times the weight of metal particles which it could theoretically float if the whole phenomenon was due to the low specific gravity of the oil.

It was recognized that the process was not a simple oil flotation process, but that the flotation was due to both oil and gas bubbles. The oil has an "affinity for the sulphide mineral in the presence of water." An oil film forms on the surface of the particle and around this a bubble of gas forms on account of "the affinity of the gas for the oil in the presence of water." Mr. Haywood offers this suggestive comparison with a balloon: The sulphide particles correspond to the basket of the balloon, the gas bubble corresponds to the gas bag and the oil film to the string which connects the basket with the gas bag.

It appears, therefore, that in the Elmore process we have really a combination of oil and gas flotation and the recognition of this fact led necessarily to some important improvements in the process; namely, the use of acid with oil and the employment of a vacuum, as will be described later on.

We will first sketch briefly the old Elmore process, the arrangement of the apparatus being shown in Fig. 1, where 1 is the mixer, 2 is the separator for separating the gangue and water from the metal and oil, and 3 is the centrifugal machine for separating the metal particles from the oil.

The mixture of pulp and water flows through 5 into the mixer 1, while the oil is introduced through 4. The inner surface of the mixer is provided with a helix so that the pulp, water and oil mixture is conveyed from one end of the mixer to the other. At the same time agitation is accomplished by means of wooden beaters 7.

The ore-water-oil mixture flows into the separator 2. The gangue and water settle at the bottom, while the metal particles are floated by the oil to the top and flow over into 18. The gangue and water mixture is siphoned off through 9 into a second mixer, etc.

The metal particles with the oil flow from 18 through the pipe 8 into the centrifugal machine 3. The construction of the latter is as follows: 14 is a non-perforated drum with a round hole 13 at the top and a round hole 15 at the bottom, the latter being closed by the inverted funnel 16. This centrifugal machine, which revolves at a speed of 800 to 1,000 revolutions per minute, is enclosed within the stationary drum 3. 14 is filled with hot water, which assumes the form indicated in the illustration when the drum is revolved. When now the oil and metal are introduced through 8 the oil forms a thin film over the whole inner surface of the water column and creeps up-

² Concerning details, see our Vol. IV, page 49.

wards and over the rim of 14. It then flows down and collects in 17 and is drawn off through 10.

The centrifugal force acts strongly on the metallic particles suspended in the oil film and forces them out of the oil film into the water column against the drum walls. When sufficient metal particles have thus accumulated on the wall, the centrifugal machine is stopped, the funnel 16 is raised and the drum is emptied. The separation is rendered somewhat more easy by preheating the oil.

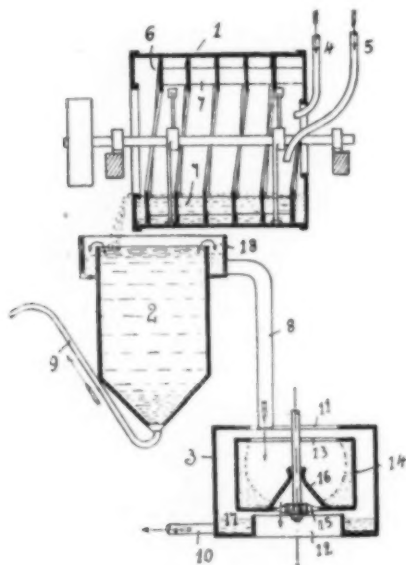


FIG. 1.—OLD ELMORE OIL FLOTATION PROCESS.

This process has been used at various plants, but was not a commercial success. The reason appears to have been that the cost was somewhat too high when the process was applied to crude ore instead of a previously half-concentrated product, and the mines where the process was installed failed to produce suitable ore or tailings for which the method could be used economically.

The process has now undergone in recent years various modifications. Not only has it been rendered continuous so as to reduce the cost of attendance to a minimum, but a considerable saving of oil has also been effected.

Since the floating effect is due not simply to the oil, but to the combined action of oil and gas bubbles, it was found advisable to produce gas bubbles artificially. By mechanically introducing small jets of air, bubbles of too large a size are produced to be efficacious. It was, therefore, found necessary to generate the gas bubbles in the pulp itself. This may be done either electrolytically or by adding sulphuric acid, generating CO_2 and H_2S gas from the carbonates and decomposed sulphides of the ore (as in the ordinary flotation processes). The latter method has been found most convenient and inexpensive.

The sulphuric acid added serves two purposes. First, it has a cleansing effect on the sulphides, making them more amenable to the action of the oil; and second, it exerts a lifting effect. An attraction is caused between the mineral particles themselves, the sulphides forming spongy masses or flocks united by gas bubbles which rise and separate from the gangue, rather than in individual particles.

The lifting power of the gas bubbles is greatly enhanced by producing a vacuum in the separator. This is the second important improvement of the Elmore process and it has been carried into effect in an ingenious way so as to reduce the power cost to a minimum. The principle is diagrammatically shown in Fig. 2.

The ore is first crushed, the size of the crush being governed

entirely by what is required to free the sulphides from the gangue matter. The size is generally from 20 to 40 mesh, although in some cases as high as 150 mesh is necessary. While any method of crushing will answer, rolls are preferred. After wet crushing to the desired size the pulp is passed through a conical tank with rim launder and all unnecessary water is eliminated.

The thickened pulp is then fed by a mechanical valve feeder B (Fig. 2) into the mixer A with a suitable sized stream of 10 per cent acid and crude petroleum. Within A the mass is rapidly and thoroughly mixed by paddles revolving at 30 to 40 revolutions per minute, and escapes into the short arm C of the U copper pipe D by which it is forced by the atmospheric pressure into the inverted cone-shape dome G of the separator, which is kept under a vacuum, through pipe V connected to a vacuum pump.

The upper end of the feed-pipe D enters the center of the separator G. Within the separator the metallic particles float to the top and are carried off through the pipe N into the tank P, while L is the tailings discharge pipe leading to the tank R. The tanks P and R form water seals for the two discharge pipes N and L respectively.

It will thus be seen that the whole arrangement is that of a siphon, so that very little power is required for operating it. The discharge of the tailings pipe L is controlled by a mechanically-operated valve and the rate of flow of the pulp down the pipe L is made slightly less than the inflow through the feed-

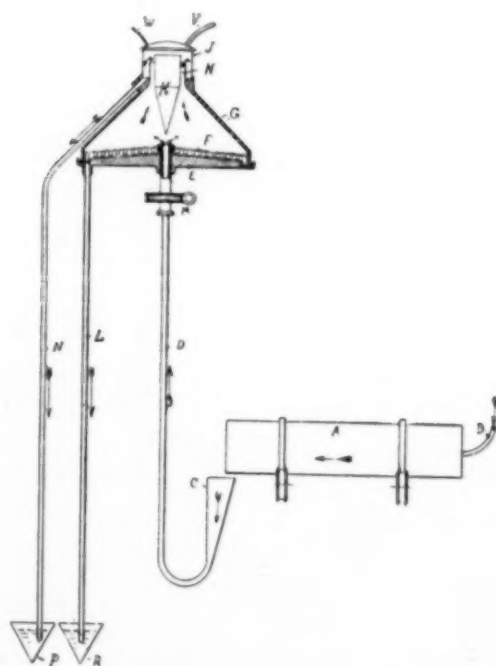


FIG. 2.—NEW ELMORE VACUUM OIL FLOTATION PROCESS.

pipe D. The result is that a small amount of the liquid overflows the lip of the annular space K, this quantity of liquid being sufficient to carry the concentrates which float on the surface down the pipe N into the tank P. The rakes F are caused to revolve by means of the worm and wheel M at a speed of 1 to 2 revolutions per minute and the angle of the rake plates is such as to cause the solid matter in the pulp to travel from the center to the periphery of the conical chamber whence the tailings continually discharge down the pipe L.

The pipe D is 25 to 30 feet long in practice, while L and N are a few feet longer. The conical separating chamber is generally 5 feet in diameter and this size treats from 35 to 45 tons crude ore in 24 hours.

It is stated that in general the operating cost on ores well

suited to the Elmore vacuum process is about the same as the cost of a modern water concentrating plant. The total cost is the sum of the following items: Crushing, crude oil, acid, labor, power, wear and tear.

The amount of crude oil required per ton of ore is 3 to 10 lb. The amount of sulphuric acid per ton of ore 1 to 20 lb. Since the operation is continuous little labor is required (according to Mr. Haywood 2 men for every 4 units of the 40-ton size mentioned above). Aside from the vacuum pump, mixer, revolving rabble and two valves, there are no moving parts, so that the item of wear and tear is almost negligible. As to the power required, this is stated to be about 0.05 hp per ton of plant capacity, hence, say, 10 horse-power for a 200-ton plant. According to Mr. B. Haywood the chief applications of the process are as follows:

While the process is primarily intended for the separation of sulphides from their gangue matter, it is sometimes possible to make very accurate separations between the sulphides themselves.

This is markedly the case with zinc blende and iron pyrites or galena and iron or copper sulphides.

The greatest efficacy of the process is naturally on those ores having heavy gangue matter, such as baryta, some hornblendes, siderite, rhodochrosite, garnet, etc., concentration of which is impossible by gravity. Its use, however, is not by any means confined to this comparatively small class of ores, since the general very high percentage of recovery frequently much more than compensates for the slight increase of cost over ordinary gravity work.

Various forms of sulphides that are difficult or impossible to concentrate with water, such as gray copper, chalcophyrite, steel galena or any galena in hard quartz, where it necessarily slimes badly, the oily minerals—such as molybdenite or graphite—are apt to do exceptionally well by the vacuum process.

Protection of Iron and Steel Surfaces by Means of Zinc.

Mr. SHERARD COWPER-COLES, the distinguished British electrochemist and metallurgist, has recently given an interesting and able résumé of the methods of protecting iron and steel surfaces by means of zinc before the Glasgow Technical College Scientific Society.

Zinc is considered to be the most suitable metal for protecting iron and steel from corrosion, because zinc is electropositive to iron and in the presence of moisture forms a galvanic couple, the zinc dissolving and the iron remaining free from oxidation. Since, however, small pieces of soot or carbon which may settle on a zinc surface will form a galvanic couple in contact with moisture and sulphurous acid (which is present in the atmosphere of all large towns), the zinc is rapidly penetrated in the neighborhood of the carbon so that it is always advisable whenever possible to cover the zinc with a coating of paint.

In all processes of galvanizing it becomes necessary to remove the scale and oxide from the iron surface. The preparation of the surface for "sherardizing" and "cowperizing," however, does not require so much care as in the case of hot galvanizing and electrogalvanizing.

Usually iron and steel to be galvanized are pickled and afterward washed. However, Mr. Cowper-Coles thinks that the operation of washing is often not carried out carefully enough and that "is one of the reasons why galvanized sheets and wire now put on the market are not so durable as the galvanized iron produced some twenty-five years ago. Another reason is that too much zinc is squeezed or wiped off."

There are now four commercial processes for applying zinc

to iron and steel. First, hot or molten galvanizing; second, electro or cold galvanizing; third, sherardizing or dry galvanizing; fourth, cowperizing or vapor galvanizing.

Hot Galvanizing.—The hot or molten process was intro-

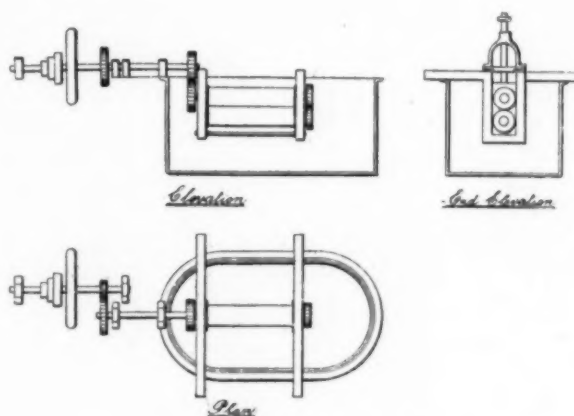


FIG. 1.—HOT GALVANIZING BATH, WITH ROLLS.

duced as early as 1846. It consists in immersing the article, after removal of all scale, oxide and grease, into a bath of molten zinc. The temperature of the zinc varies from the melting point of zinc to 800° F., according to the thinness of the coating required and the nature of the work under treatment. The higher the temperature, the thinner the coating; but the saving is not as great as might be expected, since the

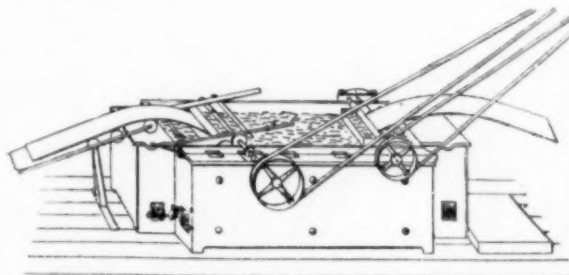


FIG. 2.—HEATHFIELD SHEET GALVANIZING MACHINE.

higher the temperature, the more zinc combines with the iron in the bath or the article, to form a zinc-iron alloy which has to be removed from the bath from time to time. The temperature of the bath is generally, therefore, kept as low as possible, and the excess of zinc is removed by shaking the articles or, if of suitable section, passing them between rollers on a bed of sand resting on the top of the molten zinc.

A typical form of rollers employed in galvanizing pots for ensuring a more even coating of zinc is shown in Fig. 1, the



FIG. 3.—CONTINUOUS PLANT FOR GALVANIZING SHEETS.

sheets being fed into the rollers by hand at the rate of about 1 ft. per second.

Messrs. Heathfield's galvanizing machine is shown in Fig. 2. Muriate of ammonia is placed in one end of the machine through which the plates enter, and sand at the other end through which the plates leave. Such a machine will turn out 12 tons or more of galvanized sheets, 14 to 30 gauge, in 10½ hours.

The galvanizing pots are made of wrought iron or steel with riveted or welded joints, and usually contain about 25 tons of molten zinc for sheet galvanizing, although they are often made large enough to contain 200 tons or more of metal when large plates or tanks have to be dealt with.

The bottom of the galvanizing pot is generally arranged on

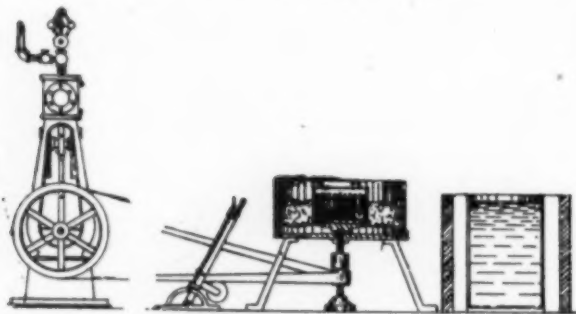


FIG. 4.—WIRE COIL GALVANIZING MACHINE.

solid masonry, the brickwork being built up around the pot so as to provide a fire space between the brickwork and the iron pot. A number of draft holes are left in the brickwork, which are used for regulating the draft by removing or replacing bricks.

Fig. 3 shows the general arrangement of a plant for galvanizing sheets continuously.

Wire is coated with zinc by passing a number of wires, arranged parallel to one another, through the bath. As the

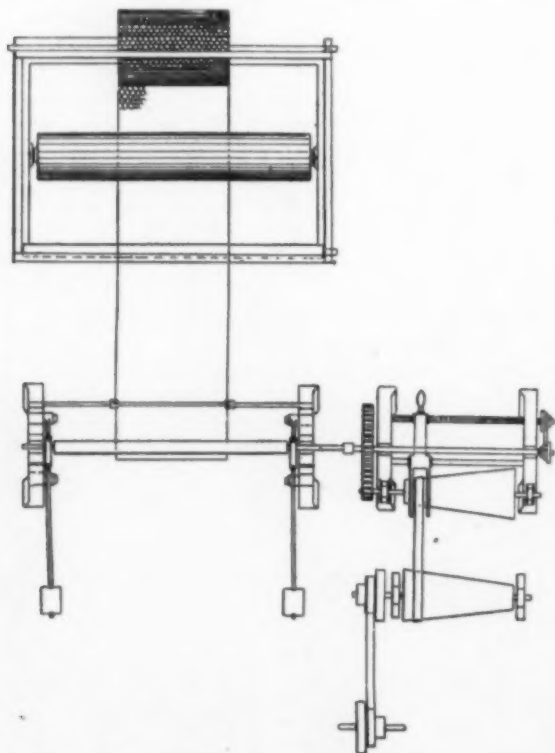


FIG. 5.—MACHINE FOR GALVANIZING WIRE NETTING.

wires leave the molten zinc they are drawn through sand or wiped with asbestos to remove some of the zinc.

Another process consists in placing the entire coil in the molten zinc and then removing some of the zinc by centrifugal force in an apparatus, as shown in Fig. 4.

Wire netting is covered with zinc by a machine such as is shown in Fig. 5, arrangements being made to vary the speed of the winding roller as the size of the roll of wire netting in-

creases. The amount of zinc required for galvanizing wire netting is considerable, due to the filling up of the interstices in the twisted wire. This difficulty has been surmounted by making wire netting with the mesh welded together by means of small electric welding machines.

One of the chief advantages of hot galvanizing, as compared with other processes, is the rapidity with which the work can be done. The soldering action due to hot galvanizing is found to be a considerable advantage when galvanizing thin tanks and such like articles, which are not caulked to make them watertight, but, on the other hand, it is responsible for a good deal of bad work which otherwise would not be passed. The process of hot galvanizing is an expensive one, due to the necessity of keeping the zinc molten continuously; the zinc, especially when overheated, rapidly dissolves the iron bath; it also dissolves off some of the iron of the articles immersed in the

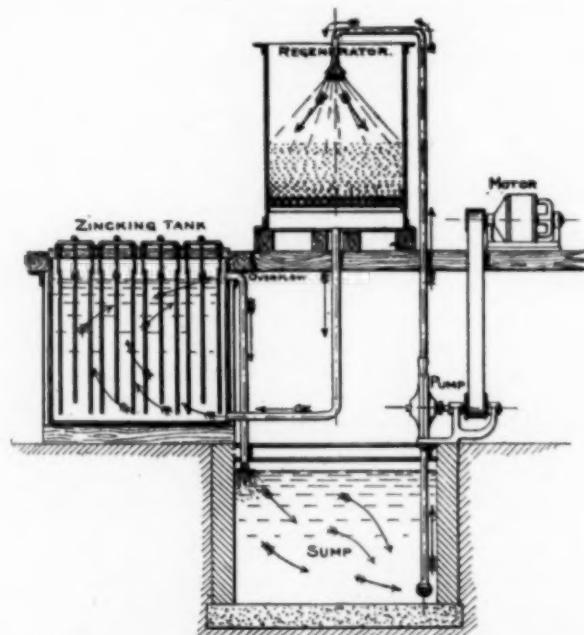


FIG. 6.—ELECTROGALVANIZING WITH REGENERATION OF ELECTROLYTE.

molten zinc, with the result that a large quantity of iron-zinc alloy is formed in the bath of molten zinc which has to be removed at short intervals.

Electrogalvanizing.—The electrolytic process of coating iron and steel surfaces with zinc has been brought in recent years to such perfection that it can be worked by any intelligent workman. The British Admiralty use the process extensively for detecting flaws in tubes, forgings, etc., before they are put into actual use. (The flaws are shown up owing to their not taking on the galvanizing.)

While in this country mostly soluble zinc anodes are used to keep up the correct amount of zinc in the bath, Mr. Cowper-Coles remarks that this is not necessary, since the electrolyte may be easily regenerated and brought up to the correct amount of zinc by passing it through a filter bed containing zinc dust and carbon. The automatic cycle of operation of electroplating, running off the electrolyte, regenerating it and returning it to the electrolytic vat, is illustrated in Fig. 6.

Lead anodes may, therefore, be used and they have the advantage that they do not need to be removed. When zinc anodes are used about 50 per cent only goes into solution and a large proportion crumbles away into the electrolyte. In this case if regeneration of the electrolyte is employed, the zinc which crumbles away from the anode is caught by the filter bed and is gradually dissolved. According to Mr. Sherard Cowper-Coles the regeneration of the electrolyte (whether zinc anodes

or lead anodes are employed) offers quite a considerable saving.

Electrogalvanizing has been very extensively applied to boiler and economizer tubes for battleships and destroyers. The tubes

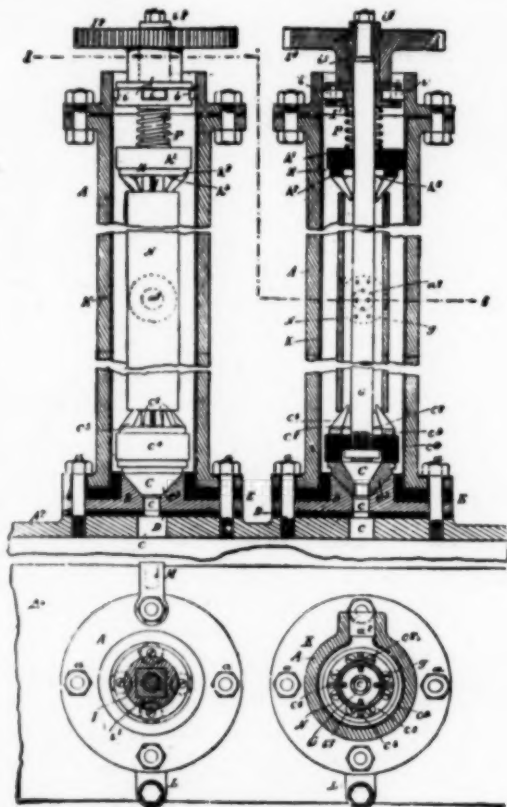


FIG. 7.—COATING TUBES INSIDE AND OUT.

are usually suspended vertically and so arranged that they can be rotated by hand.

Fig. 7 shows an arrangement for coating tubes inside and out. Small articles, such as bolts and nuts, are zinc in revolving barrels, or in such an apparatus as shown in Fig. 8.* By the addition of various organic substances zinc can now be deposited in a bright form.

[Sherardizing.]—The process of sherardizing or dry galvanizing has already been described in detail in an able and interesting article of Mr. Alfred Sang in our Vol. V, p. 187.

The iron articles to be rendered non-corrosive, after having

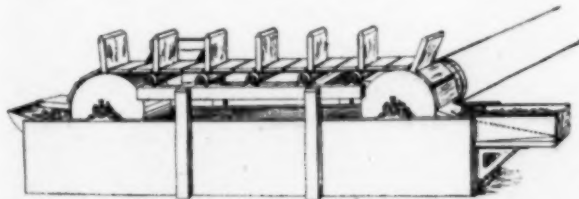


FIG. 8.—APPARATUS FOR PLATING SMALL ARTICLES.

the black scale removed, are placed in a closed iron receptacle charged with zinc dust, which can be kept stationary or rotated. The contents of the drum are heated to a temperature of 500° to 600° F. for a time varying from 30 minutes to several hours according to the nature and section of the iron to be coated. After withdrawing the drum from the heating oven or furnace, it is preferably allowed to cool, and it is then opened and the iron articles removed, when they are found to be coated with a fine homogeneous covering of zinc, the thickness of the zinc depending on the time and temperature.

*See also description of the apparatus of the U. S. Electro Galvanizing Co. on page 332 of our Vol. V.

The process is a very economical one, as the temperature is low and great economy is effected in the amount of zinc consumed, due to the fact that less zinc is required to give the same protective coating than would be necessary in the case of the hot or electrogalvanizing processes, and as all the zinc is

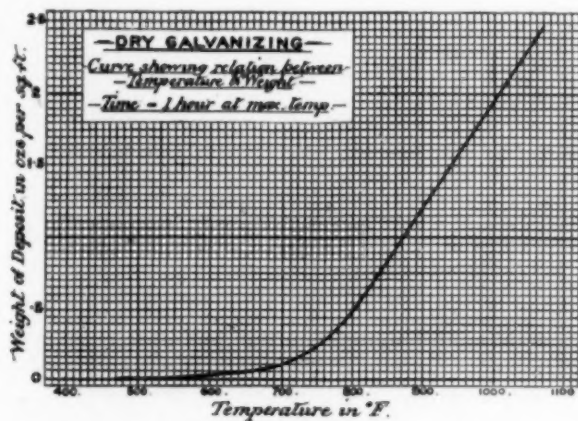


FIG. 9.—INFLUENCE OF TEMPERATURE ON THICKNESS OF ZINC DEPOSIT IN SHERARDIZING.

consumed, there are no skimmings or dross formed as in the hot process. The zinc powder used in the process is the zinc dust of commerce and must not be confused with zinc oxide.

The receptacle in which the zinc dust is placed and heated is preferably air tight, and the air is exhausted so as to prevent the formation of too much oxide. If this is not feasible, it is found advisable to add about 3 per cent by bulk of carbon in a very fine state of division. If the percentage of oxide is allowed to increase beyond certain limits, it is found that the deposits become dull in appearance, instead of having a bright, metallic lustre, although good deposits of zinc can be obtained from zinc dust containing only 35 per cent of metallic zinc. Articles

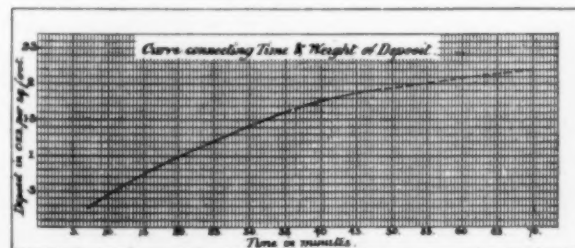


FIG. 10.—INFLUENCE OF TIME ON THICKNESS OF DEPOSIT IN SHERARDIZING.

coated with grease receive as good, if not better, coatings of zinc than those which are free from grease. This fact is of considerable importance, as it enables machined work, such as bolts, nuts, screws, etc., to be placed direct after machining into the dry galvanizing drum without any preparation or cleaning.

Temperature and time are the two factors which regulate the thickness of the zinc deposit, as shown by the two diagrams, Figs. 9 and 10. While it is always recommended that zinc surfaces exposed to the weather should be painted, sherardized surfaces admirably resist atmospheric influences without further protection.

In practice dry galvanized iron and steel are found to withstand the ordinary corrosive agents to which galvanized iron is exposed to a remarkable degree. Even after the apparent removal of all the zinc by filing or abrasion, the iron is still non-corrosive. This valuable property is doubtless due to the protective action of the zinc-iron alloy which forms on the boun-

dary line between the iron and zinc, as has been proved by micro-photographs. As dry galvanizing is effected at a very much lower temperature than hot galvanizing, the temper of steel wire is not reduced as it is in the latter process. A number of steel and iron bolts galvanized at varying temperatures, when tested for tensile strength, were found to be equal in strength to bolts which had not been dry galvanized.

The following table gives the results of tests for tensile strength upon four untreated and four sherardized specimens of steel rods:

DIMENSIONS.	AREA.	Reduction of Area at Frac. Per Cent.	EXTENSION.		Elastic Limit (Yield Point).	ON ORIGINAL AREA.		REMARKS.
			On Two Frac. Per Cent.	On Ins. at Frac. Per Cent.		Lbs. Per Sq. Inch.	Tons Per Sq. Inch.	
Untreated Steel Rod.....0.276 by 0.277	0.0774	—	1.0	—	70,358	31.41	121,676	Irregular fracture. Broke outside datum points, in jaws. Broke outside datum points, in jaws. Irregular fracture. Broke outside datum points, in jaws. Broke outside datum points, in jaws.
" " ".....0.274 by 0.274	0.0750	6.6	3.0	—	49,280	22.0	150,796	
" " ".....0.278 by 0.278	0.0772	—	3.5	—	73,987	33.03	149,408	
" " ".....0.283 by 0.268	0.0758	3.7	3.0	—	69,440	31.00	153,932	
Sherardized Steel Rod....0.279 by 0.285	0.0795	19.9	7.0	—	113,545	50.69	156,598	Steel Rods treated. Broke outside datum points, in jaws.
" " "....0.274 by 0.279	0.0764	6.7	4.0	—	102,032	45.55	148,064	
" " "....0.274 by 0.283	0.0775	15.9	6.0	—	Not Observable.		154,918	
" " "....0.276 by 0.281	0.0783	4.8	3.5	—	108,147	48.28	149,632	

The process of sherardizing has recently been turned to account for the inlaying and ornamenting of metallic surfaces, enabling results to be obtained similar to inlaying, but with the additional advantage that there is no risk of the metals finally separating.

The new process also enables a variety of effects to be obtained and a number of metals to be blended together which has hitherto been impossible, and alloys of many colors and tints to be obtained in the one operation of baking. The thickness and depth to which the metals are to be inlaid can be controlled at the will of the operator.

The *modus operandi* of the process consists in coating the article with a stopping-off composition, those portions which are to be inlaid being left exposed. The composition is about the consistency of cheese, so that it can be readily cut with a knife, the design is traced with a sharp-edge tool, and those portions to be removed are lifted and cleared away. The object thus prepared is placed in an iron box containing the metal which is to be inlaid in a powdered form.

If zinc is the metal to be inlaid, zinc dust is the powder employed. The iron box holding the powdered metal and the objects to be ornamented is then placed in a suitable baking oven and heated to a temperature many degrees below the melting point of solid zinc, which is 686° F., so that the temperature to which the zinc dust is heated is about 500° F. The time and temperature are regulated according to the thickness and depth of the inlaying which is required to be obtained and varies from a few minutes to several hours.

A useful type of furnace or baking oven for general work, such as panels, trays, etc., is shown in Fig. 11 and consists of a wrought-iron box, 8 ft. long, 4 ft. broad and 1 ft. deep. The box is half filled with zinc dust and the articles to be baked are placed in the zinc dust and well covered with it. A lid is placed on the top of the box, and over this an iron frame, balanced with counter weights, lined with fire bricks and fitted with a small central flue, which draws the heat from the burners up to the sides of the box and over the top, thus ensuring even heating of the contents of the box.

It is found that the inlaying metal, in the case of zinc, is very much harder than the brass or copper into which it is inlaid. A feature of considerable importance is that a variety of color and alloys can be obtained in the one operation of baking. Take, for example, a copper tray, which it is desired to inlay with zinc and at the same time to convert certain portions of the copper into brass. This can be done by varying the thick-

ness of the stopping-off composition, and by baking at a somewhat higher temperature than would otherwise be employed. The result is that certain portions can be converted into golden-colored brass, the other portions remaining unalloyed copper.

Copper and zinc give very marked contrasts in color. Softer contrasts can be obtained between zinc, aluminium, tin, nickel and cobalt and similar colored metals. The new process is not confined to flat surfaces, but can readily be applied to raised surfaces.

This new process of burning in and blending metals enables

a very beautiful effect to be obtained with great subtlety of color, the tones ranging from silver white zinc to yellow brass and bronzes of various shades, graduating to red copper and autumnal shades of yellows and golden browns.

A great charm about this new process of inlaying metals, and one that is unique, is that the inlay has not the sharp line of demarcation, as is essential to damascening, but a soft transition from the inlaid metal to the surrounding metal. For instance, in the case of inlaying zinc into copper it will be observed that the zinc is surrounded by a band or halo of a golden-colored alloy.

Cowperizing.—The process of cowperizing or vapor galvanizing is distinctive from all other processes inasmuch as the

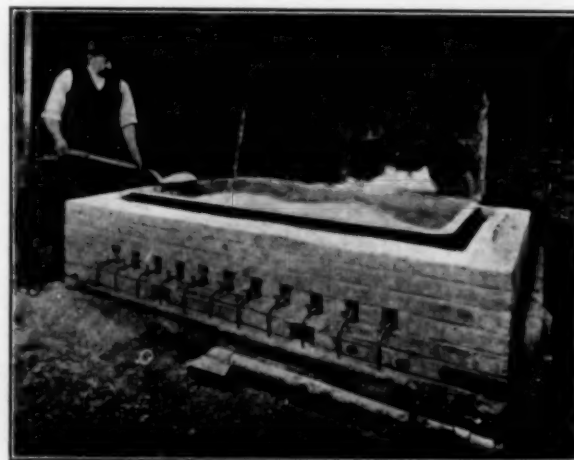


FIG. 11.—FURNACE FOR INLAYING.

articles coated with zinc are not brought into contact with molten zinc or any of its compounds, but placed in a separate chamber into which zinc vapor is passed.

A convenient form of apparatus is shown in Fig. 12. It consists of an inner drum C, composed of wire netting or gauze, in which the articles to be cowperized are placed. This cage or hollow drum is slowly rotated inside an outer cylinder B composed of wrought iron, in which the metallic zinc is heated by means of gas or an electric furnace, or any other suitable means, to a temperature sufficiently high to volatilize the zinc.

Hydrogen gas is forced into the apparatus through the tube marked A and a pilot light of hydrogen is kept burning through a small hole in the door marked D, which makes a gas-tight joint when closed. The beneficial effect of the hydrogen on the color and adhesion of the deposit is probably due to its reducing action. The apparatus for cowperizing has to be modified to suit various requirements, but in almost all cases it is necessary to move the objects to ensure their being evenly coated with zinc.

The process has also been successful when applied to decorating porcelain and metallic surfaces with a brilliant coating of metallic zinc.

The author finally sums up the comparative advantages of the different processes. He also gives tables of their cost of operation, but since these contain cost of labor and incidentals as an important item, and since the figures are based on English conditions, these tables are not reproduced here.

Electrogalvanizing Versus Hot Galvanizing.—Prof. C. F. Burgess¹, of the University of Wisconsin, has contrasted the hot and cold processes as follows: (1) In hot galvanizing the heat required is considerable, as much is lost by radiation and in bringing the articles under treatment to the requisite temperature. (2) The loss of zinc is also very considerable owing to oxidation at the surface of the zinc, and from combination of zinc with the sal-ammoniac, and with the iron of the treated articles, and through corrosion of the iron vessel. These losses amount to about 50 per cent of the zinc. The cold or electro-

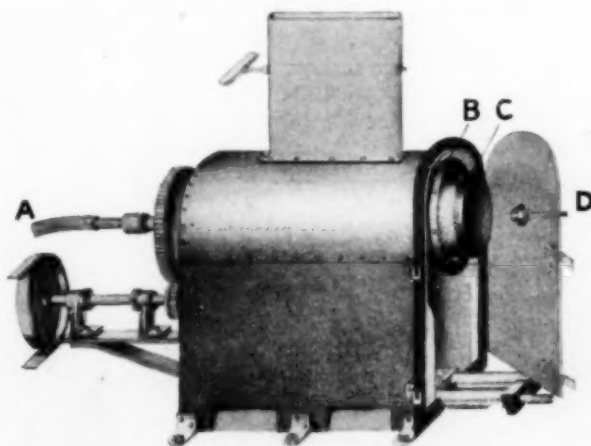


FIG. 12.—COWPERIZING APPARATUS.

lytic process, on the other hand, has the following advantages: (1) There is no loss of heat. (2) No difficulties are introduced by the high temperature of 770° F. required in the hot process. (3) No skilled workmen are requisite. (4) The loss of zinc is small. (5) For equal quantities of zinc used the electrolytic process gives a better protection. (6) The wear and tear of the plant per year is less than 10 per cent, whereas in the hot process it is some 50 per cent. (7) Articles of any size can be galvanized. (8) The electrolytic bath is always ready, while the melting of the zinc in the hot process requires time. (9) The thickness of deposit can be easily controlled. (10) Objects of art can be submitted to the cold process without any risk of damage.

Sherardizing Versus Hot Galvanizing.—The process of sherardizing has the following advantages over hot galvanizing, according to Mr. Cowper-Coles: There is no waste of zinc due to the formation of zinc-iron alloys; less zinc is required to give the same protective coating, the zinc being evenly distributed; the temperature required is lower, consequently the amount of fuel consumed is less. The labor is less, as the articles do not require to be cleansed as carefully as in hot galvanizing. No flux

is required, no dross or skimmings are formed; there is no danger of explosion, breaking of castings or distortion of thin iron work. Dry galvanized machined work does not require refitting, as the coating is evenly distributed. There is not the same reduction in tensile strength as in the case of hot galvanizing; work can be placed direct in the dry galvanizing drum from the pickling vat without drying; the process can be worked intermittently without waste; iron can be coated with zinc to any desired thickness, and the whole of the zinc is consumed without waste. The depreciation of plant is small compared to hot galvanizing.

Comparison of Hot, Cold, Galvanized and Sherardized Surfaces

—Surfaces obtained by hot and cold galvanizing and sherardizing are different in each case, but they can readily be distinguished by anybody conversant with the three processes. In case of hot galvanizing the surface is spangled, or if not spangled has the appearance of cast metal. In the case of cold galvanizing the surface is free from spangles and has a matted or frosted surface, uniform if the work has been well executed. Sherardizing is again distinctive from the two former processes; the general appearance resembles more that of cold galvanizing than hot galvanizing, but is more lustrous and metallic, and is uniformly distributed over the whole surface, which is not the case with the hot and cold galvanizing processes. The sherardizing process, although similar to cold galvanizing, is also similar to hot galvanizing in other respects, inasmuch as the zinc alloys with the iron and forms a protective zinc-iron alloy intermediate between the zinc coating and the underlying metal.

The copper sulphate test, known as Preece's test, is not applicable as a comparative test of the thickness of zinc applied by the sherardizing or electro-zincing process for the following reasons: On applying Preece's tests to sherardized and hot-galvanized articles coated with an equal thickness of zinc the former requires from three to four times the number of immersions which suffice to remove the zinc from the latter; when hot-galvanized articles are placed in a saturated solution of copper sulphate the copper is precipitated to a loose form, but when sherardized or electro-zinc articles are similarly treated the copper adheres firmly to the zinc and no fresh surface is exposed, apparently due to the deposit of zinc applied by the electro-zincing and sherardizing processes having a fine matted surface.

It would appear from these observations that the apparently great resistance to corrosion of sherardized iron when subject to Preece's test is due to the protection of the zinc by the deposited copper; so experiments were made with a solution of ferric sulphate, which dissolves zinc without forming a precipitate on the zinc coating.

To test this known areas of sherardized and hot-galvanized plate were exposed to the action of ferric-sulphate solution for an equal period and the amount of ferrous salt formed by the reducing action of the zinc determined. The column headed "Weight of Zinc Dissolved" shows the relative corrosion.

Nos.	Sample.	Zinc per sq. ft.	
		grams.	Weight of zinc dissolved grams.
1.	Sherardizing	26.908	0.080
2.	Sherardizing	26.908	0.074
3.	Sherardizing	22.93	0.057
4.	Hot-galvanizing	22.12	0.058
5.	Sherardizing	31.116	0.034

Sample No. 2 was moistened with water and allowed to dry; the oxide formed appears to protect the zinc, and this protection is more marked if water is allowed to act for a longer period than was permissible in these experiments. It will be noticed that samples 1 and 2, which had a thicker coating than samples 3 and 4, dissolved to a greater extent than samples 3 and 4, which had practically the same weight of zinc coating, dissolved.

Sample 5 was sherardized copper and, although the zinc coat-

¹See also Prof. Burgess' paper on this subject in our Vol. III, p. 17.

ing was the heaviest, yet the corrosion was in this case the least. On testing this sample with copper sulphate solution, as in the other cases, instead of a brown precipitate of copper, a bright, metallic deposit was obtained, and no further action seemed to occur.

From the result of these experiments and other tests, it may be concluded that sherardizing offers equal and probably greater resistance to ordinary corrosive influences than hot galvanized iron coated with an equal weight of zinc, and the resistance is considerably increased by repeated moistening and drying, so as to form an oxide which renders the surface less soluble.

From experiments made to elucidate the nature of the union between a sherardized coating and the base metal in comparison with the electrogalvanizing method and from microphotographs of the etched joint produced the author concludes that in the case of hot galvanizing and sherardizing there is not merely mechanical adhesion between the coating and the base, but that an alloy of zinc and iron is actually formed at the surface of junction, and this film of alloy is less liable to corrosion than is the iron itself.

Cowperizing Versus Hot Galvanizing.—The cowperizing process is especially suitable for coating articles with threads, or which require to be accurately fitted, as the zinc is evenly distributed. The zinc combines with the surface of the iron, forming a rustless zinc alloy. The articles on removal from the cowperizing drum do not require any cleaning or washing. The process is cheaper than hot galvanizing, as there is less handling and the amount of heat required is small, as only a small quantity of zinc has to be heated.

The process, like sherardizing, is not suitable for coating articles which are dependent on the soldering action of molten zinc to make them watertight, or when the spangled effect obtained by hot galvanizing is an essential condition. The whole of the zinc is consumed; there is no dross or residue formed. The process can be quickly started, as there is no large quantity of metal to get hot.

"Electro zincing, sherardizing and cowperizing all have a large field of application and will greatly increase the amount of iron coated every year with zinc, as each process has its own peculiarities which render it more suitable than any other process for specific purposes, and in many cases enables iron objects to be coated with zinc, which could not be done by the process of hot galvanizing."

Metallurgical Calculations.

By J. W. RICHARDS,

Professor of Metallurgy in Lehigh University.

The Metallurgy of Zinc—II.

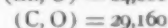
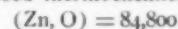
REDUCTION OF ZINC OXIDE.

Since metallic zinc boils under atmospheric pressure at 930° C., and carbon does not begin to reduce zinc oxide until 1033° is reached, the zinc reduced is necessarily obtained in the state of vapor. To make the reaction proceed fast, a temperature of the charge inside the retorts of 1100° to 1300° C. at the end is necessary. A large amount of heat is absorbed in the reaction, which must be supplied as a current or flow of heat through the walls of the retort in order to keep the reaction and reduction going. Since the walls of the retort are fire-clay, averaging 4 centimeters (1.6 inch) thick, there must be a considerable difference of temperature (temperature head) between the inside and outside of the retort in order to keep up the flow of heat inward.

THERMOCHEMICAL CONSIDERATIONS.

The heat of formation of zinc oxide, at ordinary temperatures, is 84,800 Calories for a molecular weight, 81 kilos, of oxide, as determined by thermochemical experiment. This means that cold zinc uniting with cold oxygen, and the hot product cooled down to the same starting temperature, results

in the evolution of heat stated. Cold carbon uniting with cold oxygen to cold carbonous oxide, C O, gives similarly 29,160 Calories per molecular weight, 28 kilograms, of gas formed. These facts are expressed thermochemically as



The equation of reduction becomes



$$-84,800 \quad + 29,160 = -55,640$$

This means that if we start with cold zinc oxide and cold carbon, the heat absorbed is 55,640 Calories ending up with cold products, zinc and CO at ordinary temperatures.

What is actually done in practice, however, is to first heat the Zn O and C to a high temperature, at least to 1033°, and then to supply the heat of the reaction at that temperature, producing zinc vapor and CO gas, both at 1033°. The process of reduction, therefore, resolves itself plainly into two steps: (1) heating the charge up to the reacting temperature, (2) supplying the latent heat of the chemical change at that temperature.

HEATING UP THE CHARGE.

The charge usually contains more carbon than is theoretically necessary for reduction of the zinc oxide, because some is consumed by air in the retort, some in reducing iron oxides, and some is left behind unused; it is cheaper to lose carbon than unreduced zinc oxide. However, making the calculation on the theoretical amount of carbon only (anyone can modify it for any excess of carbon used in any specific case) we need to know the heat necessary to raise these reacting substances to the temperature of reaction.

The heat in 1 kilogram of zinc oxide at various temperatures was determined in the writer's laboratory as 0.1212 + 0.0000315 t². The heat in 1 kilogram of carbon at temperatures above 1000° is represented by 0.5t - 120. We have the heat necessary to raise our 81 kilograms of Zn O and 12 kilograms of carbon to 1033° as

$$\text{Zn O: } 0.1212 (1033) + 0.0000315 (1033)^2 \times 81$$

$$= 159 \times 81 = 12,879 \text{ Cal.}$$

$$\text{C: } [0.5 (1033) - 120] \times 12 = 396 \times 12 = 4,752 "$$

$$\text{Sum} = 17,631 "$$

This quantity represents the heat per 81 of oxide and 12 of carbon, or 93 of charge containing 65 of zinc. Per 1000 kilograms of oxide, containing 800 kilos of zinc, it would be

$$17,631 \times \frac{1000}{81} = 217,670 \text{ Calories.}$$

It should be noted that this quantity is actually proportional to the weight of the charge, or plus reducing agent, and independent of the amount of zinc in it. A ton of poor ore will practically require as much heat to bring it up to the reaction temperature as a ton of rich ore, so that these costs are proportional to the weight of charge treated and not to the weight of zinc it contains.

If the charge is heated electrically, the amount of electric power needed for heating up can be calculated, assuming an average loss of 10 to 30 per cent. of the total heat by radiation and conduction from the furnace.

Illustration:

A retort contains 300 kilograms of charge mixture and is heated electrically to 1033°, the reaction temperature at an efficiency of 75 per cent., by an electric current of 250 horse-power. How long will the heating-up period last?

Solution:

Heat needed in the charge, $217,670 \times 0.3 = 65,300$ Calories. Heat to be supplied $= 65,300 \div 0.75 = 87,070$ Calories. Power applied supplies $635 \times 250 = 158,750$ Calories per hour. Time required $87,070 \div 158,750 = 0.55$ hour = 33 minutes.

If the charge is heated by furnace gases outside the retort we must take into consideration that the fire-clay is a poor conductor, that the rate of transmission of heat through the walls falls off as the charge becomes hot, and that the outside sur-

face of the retort must be kept well above 1033° in order to get the charge to that temperature in a reasonable time. The conductance of fire-clay for high temperature is 0.0031; that is, 0.0031 gram Calories pass through each square centimeter per second, if one centimeter thick, per 1° C. difference of temperature. When the charge is cold, the inner surface of the retort may be reduced in temperature to a low red heat, say, 500° , and towards the end of the heating-up period its temperature becomes at least 1033° , while the outer surface is kept continually at 1200° , let us say, by the furnace gases. During the heating-up period there is a difference of temperature producing heat flow of 700° , for a short time, down to, say, 200° , which we may average up as 300° heat difference. The heat conductivity of the material of the retort and the thickness of its walls have everything to do with the rate at which the heat can get through and the charge be heated up to the reaction temperature.

Problem 130.

An oval zinc retort is 130 centimeters long, 30 centimeters diameter outside, one way, and 15 centimeters the other, and its walls are 3 centimeters thick. It is charged with 30 kilograms of ore and 12 kilograms of reduction material. The temperatures of the charge in the retort and of the gases outside the retort were as follows:

	In Retort °C.	Outside °C.	Difference °C.
At starting.			
In 0.5 hour	350	1067	717
" 1.0 "	600	1067	467
" 1.5 hours	781	1067	286
" 2.0 "	814	1100	286
" 2.5 "	869	1100	231
" 3.0 "	924	1110	187
" 3.5 "	957	1155	198
" 4.0 "	935	1166	231
" 4.5 "	935	1138	203
" 5.0 "	946	1144	198
" 5.5 "	946	1155	209
" 6.0 "	979	1166	187
" 6.5 "	1001	1177	176
" 7.0 "	1034	1177	143

Average, 319°

Take 159 Calories per kg. as the heat required to bring the ore to 1034° and 396 for the reduction material.

Required:

The average heat conductivity in C. G. S. units of the material of the retorts, in the range given, assuming the inner surface of the retort to be at the same temperature as the charge.

Solution:

Heat passing through the retort walls in 7 hours:

$$159 \times 30 = 4770 \text{ Calories}$$

$$396 \times 12 = 4752 \text{ "}$$

$$9522 \text{ "}$$

$$\text{Per second} = 0.378 \text{ "}$$

$$= 378 \text{ gram. cal.}$$

Surface of retort:

$$\text{Periphery } \sqrt{30 \times 15} \times 3.14 = 66 \text{ cm.}$$

$$\text{Area of sides, } 66 \times 130 = 8580 \text{ sq. cm.}$$

Heat passing through each square cm. per second

$$378 \div 8580 = 0.044 \text{ Cal.}$$

Heat passing per 1° difference

$$0.044 \div 319 = 0.00014 \text{ Cal.}$$

Since thickness is actually 3 centimeters, conductance in C. G. S. units is:

$$0.00014 \times 3 = 0.00042.$$

Correction:

This coefficient of conductance is far too low. The reason is that the inner temperature, the temperature of the charge, is always lower than the temperature of the inside surface of the retort, and the difference of temperature between the outer and

inner walls of the retort must have been far less than the average, 319° . If we take the coefficient of conductance as determined by experiment for firebrick, viz., 0.0031, then the average difference of temperature between the inner and the outer walls of the retort would be:

$$319 \times \frac{0.00042}{0.0030} = 45^{\circ} \text{ C.}$$

This is more likely than that the conductance of the retort material should be so extraordinarily low. In fact, we are led to the conclusion that the poor heat conductivity of the charge itself is the chief obstacle to its rapid heating, and that pre-heating of the charge would be very advisable if it could be done by some of the waste heat of the gases leaving the furnace.

DISTILLATION OF THE CHARGE.

The driving off of the zinc is altogether a different operation from the heating up of the charge to the stated reduction temperature. It is an endothermic chemical operation, comparable to the boiling of water at a constant temperature, the latent heat of the chemical reaction is exactly comparable to the latent heat of vaporization. The temperature must be kept up to the temperature of reduction in order for the reaction to take place at all, and then heat-Calories must be supplied at that temperature to keep the reaction going, and the reduction proceeds *pari passu* with the quantity of heat supplied at that constant temperature. The question now is: what is the latent heat of this reaction at the reaction temperature?

We must for this purpose know the heat of formation of the substances involved, ZnO and CO, at 1033° . The method of calculating these is too long to insert here, but may be learned from the writer's book on Metallurgical Calculations, Part I, p. 51. We have the heats of formation from the elements as they exist at 1033° :

$$(\text{Zn, O})^{1000} = 112,580$$

$$(\text{C, O})^{1000} = 30,091$$

and the reaction at 1033°



$$-112,580 + 30,091 = -82,489$$

This is seen to be nearly 50 per cent. greater than the heat of the reaction calculated to ordinary temperatures, from the ordinary heats of formation as taken from thermochemical tables. The metallurgist should understand this difference, for it is one of the utmost importance in thermochemical calculations, if we want our calculations to represent actual conditions and to check up with practice.

This amount of heat is proportional to the zinc oxide reduced or to the zinc distilled from the charge, but not to the weight of ore charge itself. This requirement will, therefore, be greater, per retort full of material, the richer the charge is in zinc. It is a constant requirement for a given output of zinc, and not per given weight of ore treated.

The heat required for the reduction, therefore, as distinguished from the heating-up period, is

$$82,489$$

$$\frac{82,489}{81} = 1,018 \text{ Calories per kg. of Zn O reduced}$$

$$81$$

$$82,489$$

$$\frac{82,489}{65} = 1,269 \text{ " " " " Zn distilled off}$$

$$65$$

$$= 1,269,000 \text{ " " ton " " " "}$$

Furnishing this reduction heat, at the high temperature required, is the larger part of the heat needed in the whole process. We see that it is some 4.7 times the amount of heat needed to raise the materials to the reduction temperature.

Problem 131.

The retort charge of Problem 130 was kept at the reduction temperature for 14 hours, the temperature outside the retort being gradually raised to 1300° and the average difference in temperature between the gases and the charge being 147° , and

there being distilled from the charge in that time 18 kilograms of zinc.

Required:

The average heat conductance in C. G. S. units of the material of the retort from the above data and assumptions.

Solution:

The heat furnished in the 14 hours was as follows:

$$1,269 \times 18 = 22,842 \text{ Calories.}$$

Heat furnished per hour:

$$22,842 \div 14 = 1,632 \text{ "}$$

Heat furnished per second:

$$= 0.453 \text{ "}$$

$$= 453 \text{ calories.}$$

Heat passing each sq. c.m. of retort surface per second:

$$453 \div 8,580 = 0.053 \text{ calories.}$$

Per 1° difference:

$$0.053 \div 147 = 0.00036 \text{ "}$$

Conductance, in C. G. S. units:

$$0.00036 \times 3 = 0.00108$$

Remarks: This conductance calculates out 2.5 times as great as from the data on the heating-up period, the reason of the higher value being that the charge is at nearly uniform temperature during this reduction period, and therefore the difference between the temperature taken in the middle of the charge and the true temperature of the inner walls of the retort is less than before, and the error from this source less. If we make the same assumption as in the correction to the previous problem, i. e., take the conductance of the retort material as 0.0030, the difference in temperature of the outer and inner walls of the retort during this period would calculate out

$$\frac{147 \times 0.00108}{0.0030} = 53^\circ$$

while the center of the charge was then $147 - 53 = 94^\circ$ cooler, on an average, than the inner wall of the retort from which it was deriving its heat.

These figures appear reasonable, and the writer would conclude, from the data so far available, that the conductance 0.003 probably represents a good approximation to the correct value for zinc retort material, but that, in order to use it, we should have more experimental data as to the average difference between the temperature of the inner walls of the retort and the temperature of the charge at various points in the retort.

Problem 132.

A zinc ore containing 50 per cent of zinc is mixed with 40 per cent of its weight of small anthracite coal, and retorted in a Belgian furnace. The recovery of zinc was 82 per cent of the zinc content of the ore. The consumption of anthracite to heat the furnace was 2.25 tons per 10 of ore. The anthracite contained 90 per cent of carbon and 10 per cent of ash, and had a calorific power of 7500.

Required:

(1) The total consumption of fuel per 1000 of zinc obtained.

(2) The efficiency of transfer of heat from the furnace gases to the charge.

Solution:

(1) Zinc charged, per 1000 of zinc obtained

$$1000 \div 0.82 = 1220$$

Ore charged, per 1000 of zinc obtained

$$1220 \div 0.50 = 2440$$

Coal charged with ore

$$2440 \times 0.40 = 976$$

Coal burned in grate

$$2440 \times 2.25 = 5490$$

Total coal used, per 1000 of zinc obtained

$$976 + 5490 = 6466$$

(2) Calorific power of coal burned

$$5490 \times 7500 = 41,170,000$$

(1)

Heat required to raise ore to reduction point

$$2440 \times 159 = 387,960$$

Heat required to raise fuel to reduction point

$$\text{Ash } 98 \times 159 = 15,580$$

$$\text{Carbon } 878 \times 396 = 347,690$$

Total to raise charge to reduction point

$$= 751,230$$

Heat absorbed in distilling away 1000 of zinc

$$= 1,269,000$$

Total heat utilized

$$= 2,020,230$$

Thermal efficiency of utilization of the fuel

$$\frac{2,020,230}{41,170,000}$$

$$= 0.049 = 4.9 \text{ per cent.}$$

(2)

Problem 133.

Natural gas from Iola, Kan., has the following composition:

CH ⁴	93	per cent.
H ²	2	"
CO	1	"
C ² H ⁴	1	"
N ²	3	"

It is used in a zinc retort furnace, at an efficiency of transfer of heat to the charge of 4.9 per cent., as calculated in Prob. 132, working a charge which absorbed 2,000,000 calories per 1000 kilograms of zinc distilled off.

Required: The volume of natural gas required to be used to displace the anthracite fuel used per 1000 kilograms of zinc produced. The cubic feet of gas per 1000 lbs. of zinc produced.

Solution:

Calorific power of the gas

$$\text{CH}^4 \quad 0.93 \times 8623 = 8009$$

$$\text{H}^2 \quad 0.02 \times 29030 = 581$$

$$\text{C}^2\text{H}^4 \quad 0.01 \times 14365 = 144$$

$$\text{CO} \quad 0.01 \times 3062 = 31$$

$$8765$$

This result may be called calories per cubic meter of gas, or ounce-calories (1° C.) per cubic foot, according to whether it is desired to work in metric units or the English system.

Cubic meters of gas required, per 1000 kilograms of zinc produced:

$$\frac{2,000,000}{8765} \div 0.049 = 4,700 \text{ cubic meters.}$$

Per 1000 lbs. of zinc produced:

$$\frac{2,000,000}{0.049} \times 16 \div 8765 = 75,200 \text{ cubic feet.}$$

Electric Smelting of Zinc Ores.

In an interesting communication to the American Electrochemical Society (Vol. XII, p. 117), Gustave Gin calculates the electric power necessary for the smelting of several varieties of zinc ore, assuming that the zinc vapor and other gases pass out of the furnace at the usual high temperature—1200° C.

Mr. Gin makes his calculations of heat required on the basis of molecular weight of zinc compound reduced; that is, for 81 parts of ZnO and 65 parts of Zn. Calculating to 1200°, he finds the heat in the products Zn and CO to be

$$\text{In 65 parts Zn} \dots\dots\dots 26,720 \text{ Cal.}$$

$$\text{" 28 " CO} \dots\dots\dots 8,280 \text{ "}$$

$$\text{Sum} \dots\dots\dots 35,000 \text{ Cal.}$$

whereas we calculate for the same quantities

$$\text{In 65 parts Zn} \dots\dots\dots 37,695 \text{ Cal.}$$

$$\text{" 28 " CO} \dots\dots\dots 8,945 \text{ "}$$

$$46,640 \text{ Cal.}$$

The difference between these numbers is principally in the latent heat of vaporization of zinc, which Gin assumes as

15,370 Calories, but which by a method of evaluation used by the writer figures out 27,670 Calories, and almost exactly the same value has been obtained by a different method by W. McA. Johnson.

Assume then, that we start with cold materials and end with the products of the reaction leaving the retort at 1200°, the sum total of usefully applied heat is the heat of the reaction calculated at ordinary temperature plus the sensible heat of the products at 1200°, or

Heat absorbed by reaction, ordinary temperature.. 84,800 Cal.
Heat in necessary products, at 1200°..... 46,640 "

Total 131,440 Cal.

To this must be added the sensible heat in the residue left in the retort, to get the total heat which has been applied to the charge. If the charge were pure ZnO, with the theoretical amount of carbon, the residue would be *nil*, but in practice there is always a residue of gangue with unused carbon.

Mr. Gin, having calculated the heat requirement on the above basis, then makes his calculations of power required per ton of ore smelted in the following ingenious way: The weight of each component of 1000 kg. of ore is divided by its molecular weight, and thus the number of molecular weights of material in one ton of ore determined, which, so to speak, gives a kind of chemical formula for the ore. An example will make this clear.

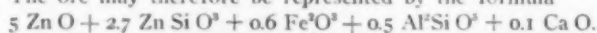
Composition of a calcined calamine zinc ore:

ZnO	40.50 per cent.
ZnSiO ³	38.07 "
Fe ² O ³	9.60 "
Al ² SiO ⁵	8.10 "
CaO	2.80 "

If we divide the weight of each compound present by its molecular weight we get the relative number of molecules present in the ore, and the formula for the ore:

KG. IN 1000 KG.	
Zn	405 ÷ 81 = 5.0 molecular weights.
Zn Si O ³	380.7 ÷ 141 = 2.7 " "
Fe ² O ³	96. ÷ 160 = 0.6 " "
Al ² SiO ⁵	81 ÷ 162 = 0.5 " "
Ca O	28 ÷ 56 = 0.1 " "

The ore may therefore be represented by the formula



Assuming that the Fe²O³ becomes Fe Si O³, and that there is to be added enough Ca O to form Ca Si O³ with the rest of the Si O³ of the zinc silicate, we will need 1.5 Ca O to do it, and since there is 0.1 Ca O present, 1.4 Ca O must be added, which represents $1.4 \times 56 = 79$ kg. of Ca O. To form CO with the oxygen combined with the zinc and iron, reducing the latter to Fe O, will require:

for 5 Zn O	5.0 C.
" 2.7 Zn Si O ³	2.7 "
" 0.6 Fe ² O ³	0.6 "

Sum 8.3 C. = 100 kg.

If we use twice the theoretical amount of carbon needed for reduction we have the following balance sheet; weights in kilograms being enclosed:

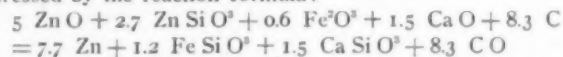
ORE.	ADDED.	CHARGE.
.5 Zn O (405)5 Zn O (405)
2.7 ZnSiO ³ (381)	2.7 ZnSiO ³ (381)
0.6 Fe ² O ³ (96)	0.6 Fe ² O ³ (96)
0.5 Al ² SiO ⁵ (81)	0.5 Al ² SiO ⁵ (81)
0.1 Ca O (28)	1.4 Ca O (79)	1.5 Ca O (79)
	16.6 C (200)	16.6 C (200)

The distribution of this charge will be as follows:

CHARGE.	GASES.	RESIDUE
5 Zn O (405)	5 Zn (325)	..
2.7 ZnSiO ³ (381)	2.7 Zn (175)	..
0.6 Fe ² O ³ (96)		1.2 FeSi O ³ (158)
0.5 Al ² SiO ⁵ (81)		0.5 Al ² SiO ⁵ (81)

CHARGE.	GASES.	RESIDUE.
1.5 Ca O (79)		1.5 Ca Si O ³ (174)
16.6 C (200)	8.3 CO (232)	8.3 C (100)

The essential reactions involved in the reduction are expressed by the reaction formula:



The heat requirements per ton of ore treated may therefore be figured out as:

Decomposition of	7.7 Zn O = 84,800 × 7.7 = 652,960
" "	2.7 Zn Si O ³
	into Zn O and Si O ³ = 10,000 × 2.7 = 27,000
Decomposition of 0.6 Fe ² O ³	
	into Fe O and O = 64,200 × 0.6 = 38,520
Heat absorbed in decompositions	= 718,480
Formation of 1.2 Fe Si O ³	
	from Fe O and Si O ³ = 8,900 × 1.2 = 10,680
Formation of 1.5 Ca Si O ³	
	from Ca O and Si O ³ = 17,850 × 1.5 = 26,770
Formation of 8.3 CO	= 29,160 × 8.3 = 222,030

Heat evolved in formation heats 259,480

Net heat of chemical reactions absorbed	459,000
Sensible heat in	7.7 Zn = 46,640 × 7.7 = 359,130
" "	8.3 CO = 8,945 × 8.3 = 74,240
" "	8.3 C = 5,760 × 8.3 = 47,810
" "	1.2 Fe Si O ³
	0.5 Al ² Si O ⁵
	1.5 Ca Si O ³
	= 413 kilograms = 460 × 413 = 200,000

Sensible heat in products and residue = 681,180

Add heat absorbed in chemical reactions = 459,000

Total heat requirement of the charge = 1,140,180

Important principle: The heat absorbed in an electric furnace by chemical reactions is electric energy utilized at an efficiency of 100 per cent. It is only part of the sensible heat of the materials being treated which is lost by radiation and conduction. Heat losses by radiation and conduction in electric furnace processes should be expressed upon the total energy of the current *diminished by the heat absorbed in chemical reactions*, and not upon the total energy of the current. This principle has been expressed most clearly by Mr. F. T. Snyder, of Chicago, the pioneer electric furnace zinc metallurgist of America.

Expressed thus, electric furnaces give higher efficiencies the greater the absorption of heat in chemical reactions taking place within them. In mere physical processes, such as melting, electric furnaces on a large scale give 75 per cent. net efficiency, with 25 per cent. loss by radiation and conduction. In the above case, figured out for 1000 kg. of zinc ore, 40 per cent. of the total heat requirement is absorbed as chemical heat at 100 per cent. efficiency, and 60 per cent. is needed as sensible heat. This 60 per cent. then represents the net sensible heating effect, and the loss of heat by radiation and conduction must be calculated as one-third of this quantity, and not one-third of the total net heat requirement. We therefore have:

Net heat requirement for chemical reactions.....	459,000
" " " sensible heat	681,180
Loss by radiation and conduction.....	227,060
Gross heat requirement of the furnace.....	1,367,240
Kilowatt hours of current required	

1,367,240

860 = 1530

860

Mr. Gin's figures have been modified by the writer, as explained above, and anyone consulting his paper on this subject

may make similar modifications to the other cases cited in that paper.

Mr. F. T. Snyder, of the Canada Zinc Company, at Vancouver, B. C., has operated the first practical electric zinc smelting furnace in America. Treating mixed lead and zinc ores, Mr. Snyder uses the furnace and process protected in his United States patents of July 2, 1907. (See this journal, August, 1907, p. 323; December, 1907, p. 489.) The lead is obtained liquid, and the zinc also condensed to the liquid state before leaving the furnace proper, the heat of condensation being partly absorbed by water cooling the condensers and partly by the descending charges, which carry it back into the focus of the furnace. Under these circumstances, Mr. Snyder reports that he has attained unexpectedly low results as to power requirement. It will be recalled, that in discussing the question of the electric smelting of zinc ores in general, and Mr. Gin's process in particular, we assumed the zinc vapor and CO gas to escape from the furnace at a minimum of 1033° C. If, as in Mr. Snyder's furnace, they escape at about 500°, the zinc liquid, the heat in these hot products is reduced very considerably, particularly that in the zinc. Mr. Snyder states in discussing Mr. Gin's paper (*loc. cit.*) that he has smelted pure zinc oxide at an expenditure of 1050 kilowatt hours per 1000 kilograms of oxide. Let us see how this coincides with the theoretical figures:

	Calories.
Heat value of 1050 k.w.-hours	$= 1050 \times 860 = 903,000$
Heat of the chemical reactions, assumed to take place	
at ordinary temperatures	$1000 \times 687 = 687,000$
Sensible heat in products, at 500°	
Zinc: $800 \times 80 = 64,000$	
CO: $342 \times 132 = 52,000$	
	$= 116,000$

Leaving, by difference, for radiation, conduction and cooling water

= 100,000

Mr. Snyder does not give details as to the exact working of his furnace, but his claim to reduce a ton of zinc ore with 1000 k.w.-hours is seen to be a possibility, if he can remove the zinc as liquid zinc from the furnace, not lose too much heat in cooling water, and get most of the heat of condensation of the zinc vapor usefully returned by the descending charges into the working focus of the furnace.

Snyder's furnace, working as claimed, would show a useful efficiency of

$\frac{803,000}{903,000} = 0.88 = 88 \text{ per cent.}$

on the current used, with 12 per cent. losses. The real heat losses, however, should be expressed upon the energy used less that absorbed in chemical reactions, or as 100,000 loss on 216,000 used for sensible heat,

$\frac{100,000}{216,000} = 0.46 = 46 \text{ per cent.}$

This shows that the furnace loses by radiation, conduction and cooling water 46 per cent. of the energy developed as sensible heat in the furnace, but the latter item is only 24 per cent. of the total energy applied to the furnace.

In short, about three-quarters of all the energy consumed by the furnace is utilized in the heat of the chemical reactions produced; of the other one-quarter, half of it is represented by the sensible heat of the products leaving the furnace and half is lost by radiation, conduction and cooling water.

Wiley Dinner.—A complimentary dinner was given to Dr. Harvey W. Wiley on April 9 at the Hotel Astor, New York City, to celebrate the twenty-fifth anniversary of his connection with the Department of Agriculture. Some 200 chemists participated. Mr. William J. Schieffelin acted as toastmaster.

Separating Appliances.

BY OSKAR NAGEL, PH.D.

Filter Presses.

The most important appliance for separating solids from liquids by filtration is the filter press. In its simplest form it consists essentially of a series of chambers, formed either by recessed plates (Fig. 1) or by flush plates with frames between them (Fig. 2). These plates and frames have a lug projecting from each side, the lugs resting on a pair of parallel bars.

One end of each of these bars is secured to the front or head of the press. The plates and frames rest upon them, and

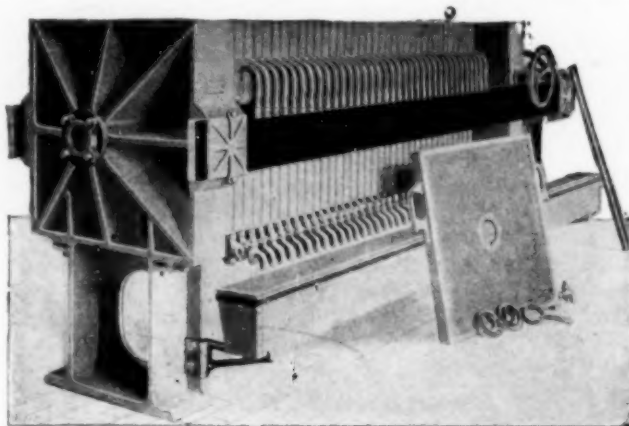


FIG. 1.—RECESSED FILTER PRESS.

the back or follower is forced up against the plates by means of a heavy screw or hydraulic plunger in a yoke or screw standard to which the rear end of each bar is secured.

Over the surface of each plate is stretched the filtering medium, either cloth or paper.

The material to be filtered is pumped in through a channel in the head of the press and is distributed over the surface of the filtering medium, the liquid passing through and out of cored channels in the plates, while the solid material is retained on the surface of the filtering medium, gradually filling the chambers until a solid cake is formed in each. The press is then opened and the cakes are removed. The press is then closed again and the operation repeated.

In the "recessed filter press" the material to be filtered is pumped in through a channel in the center of the head and the filtrate runs out through cocks or bibs provided near the bottom on the sides of the plates. It runs into a trough, which is connected by piping with a suitable receptacle. The advantage of using cocks is that in case a single cloth breaks, that plate may be shut off and the filtration continued.

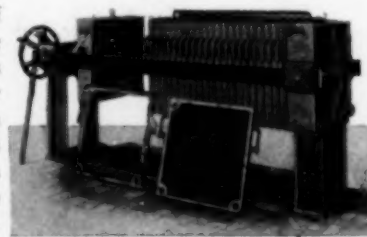


FIG. 2.—FLUSH PLATE AND FRAME FILTER PRESS FOR GENERAL CHEMICAL PURPOSES.

The "flush plate and frame filter presses" have the inlet on one side of the head, instead of in the center, and the material passes along through holes in the handles of the plates and frames. This cored channel is kept from leaking by using rubber collars inserted in the holes of each plate and frame. In this type no holes have to be cut in the cloths.

The thickness of the cake varies from 1 to 6 inches; with materials that are difficult to filter thin cakes have to be produced.

For freeing the cake in the filter press from adhering liquid

particles, most filter presses are provided with a washing arrangement, by means of which the cakes can be washed with water, diluted acids, lyes, benzene, alcohol, etc. For this purpose two extra channels are provided in every filter plate and chamber respectively. The washing liquor enters through one channel and leaves through the other. The former is con-

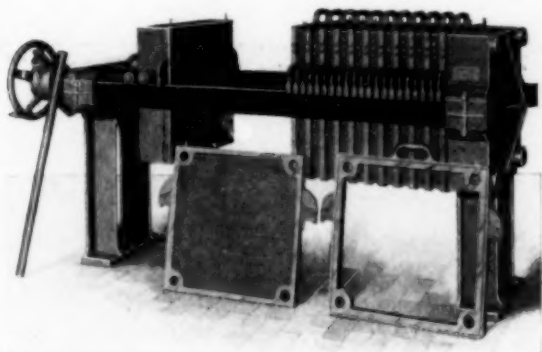


FIG. 2A.—FILTER PRESS FOR VARNISH WORKS.

ducted to every second plate only, the cocks of which are closed and the washing liquor passes behind the cloth (i. e., behind the cake) into the press, goes through the first filter cloth, then through the cake and the second filter cloth and leaves finally through the other channel, carrying along the liquid displaced from the cake. The object of washing the cakes is either to produce as pure a solid as possible, or to recover all the liquid.

In certain cases the mass to be filtered has to be kept hot or cold during filtration; hence, for these purposes, channels are cast or coils provided in the plates for the circulation either of hot water or steam or of cold water or a refrigerating mixture. The connection of the channels or coils from cham-

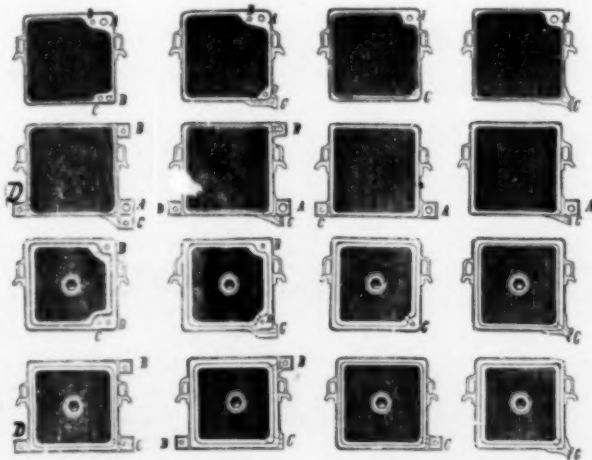


FIG. 3.—ARRANGEMENTS OF CHANNELS IN VARIOUS TYPES OF FILTER PRESSES.

ber to chamber is perfectly tight, so that the mass to be filtered cannot come in contact with the heating or cooling substance.

Heated chambers are used for materials that can be filtered only in molten state (wax, ceresine, etc.) or for liquids from which salts would crystallize out at a low temperature. Cooled chambers are used, if substances are to be separated, which solidify at low temperature only (clarification of cod liver oil, etc.).

In Fig. 3 are shown the most common arrangements of the various channels. A is the channel for introducing the material, B the washing channel, C the discharge of the filtrate

and D the discharge of the wash-water. The 16 cases shown differ from each other with respect to the following four points:

(1) The filtrate discharge outlets may be closed or open or regulable by means of cocks.

(2) The chambers may be provided with and without a washing arrangement.

(3) The method of introducing the washing liquid may be varied.

(4) It may enter through various pockets. This makes it unnecessary to cut a hole into the filter cloth, so that its life is prolonged by the use of "pocket-shaped" cloths.

The materials used in the construction of filter presses de-

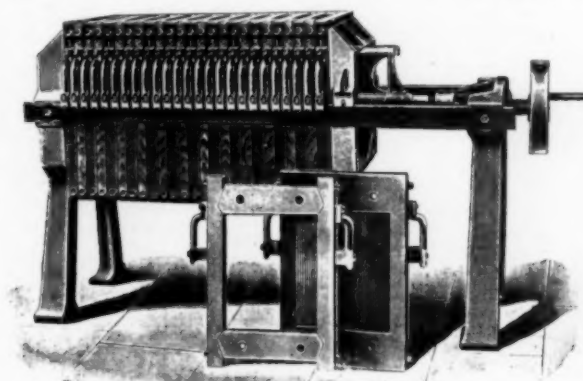


FIG. 4.—WOODEN FILTER PRESS.

pend upon the liquids to be filtered. For liquids which do not attack iron, iron is the best material. For acids, wood (Fig. 4) or hard lead is used; or the iron plates are coated with lead, tin or hard rubber. For very hot acid liquid bronze is generally used.¹

The screw-closing device as shown in Fig. 2 is generally used. The "hydraulic" closing device means simply the transformation of the end of the filter press into a hydraulic press (Fig. 5). An excellent European closing device, which is easily understood, is shown in Fig. 6.

In the recessed filter press the chambers can be tightly closed, as always two cloths are in contact, while in the flush plate and frame filter press the cloth of the plate is in contact with

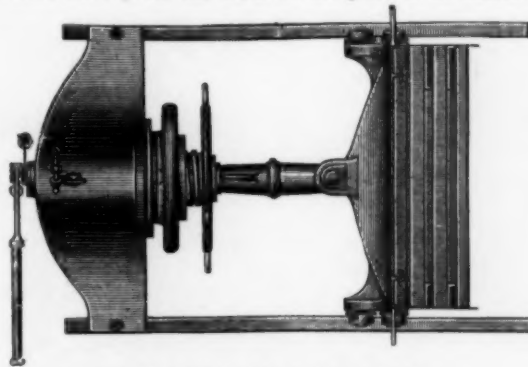


FIG. 5.—HYDRAULIC CLOSING DEVICE.

the metal frame. Hence in the latter type, the frame has to be cleaned and smoothed after every operation.

The recessed plates are stronger and have a longer life than the frames; another advantage of the recessed plate is the wide feed-opening in the center. This absolutely prevents any stopping, while in frame presses stoppages sometimes occur.

The main drawback of the recessed filter press, is the tedious and expensive way of fastening the filter cloth. This incon-

¹ See also the article by Emil Hatschek on materials used in the construction of filter presses, *ELECTROCHEMICAL AND METALLURGICAL INDUSTRY*, Vol. III., p. 220.

venience is done away with in frame presses, as here the cloth is simply laid around the chamber and connected at the bottom end by a few stitches. Another advantage of the latter type

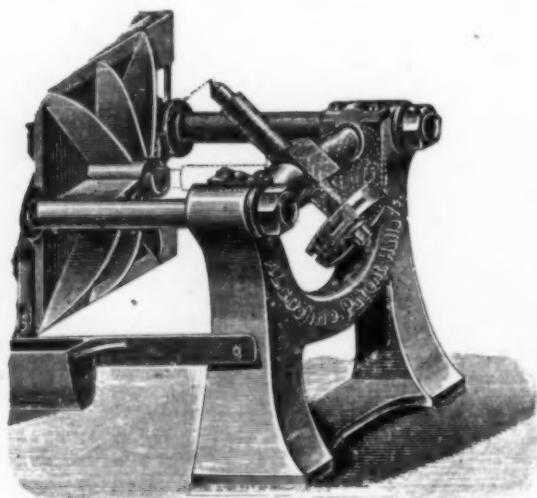
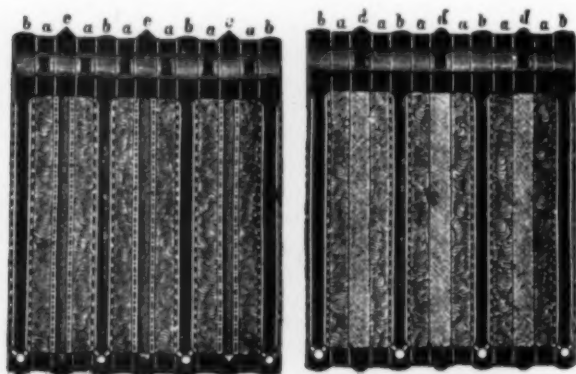


FIG. 6.—CLOSING DEVICE.

is the fact that a cake without a hole in the center is obtained.

For filtering volatile materials the filter press shown in Fig. 2 can be used, since in this type the filtrate is discharged through a pipe at one corner of the head and therefore can be kept from exposure to the atmosphere.

In some cases the filter press is covered by an air-tight sheet-iron cover, which is connected to a receiving or condensing chamber by suitable pipe connections.



FIGS. 7 AND 8.—THREE-CHAMBER PRESS.

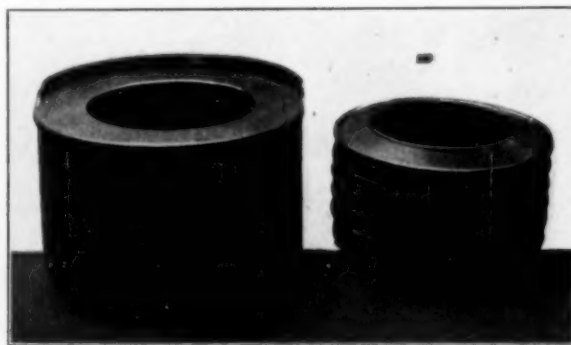
For obtaining perfectly clear solutions—which is quite difficult by the use of filter cloths—A. L. G. Dehne in Halle a. S., Germany, has constructed a so-called three-chamber-press, in which filtration is effected by sand, coal, etc., contained in the press, instead of using cloths. Fig. 7 shows the general arrangement; between two frames *a* a plate *c* is provided, while at the other sides of the frames *a* the plates *b* are arranged. The plates *b* and *c* serve for discharging the liquid during the preparation of the filter. The filtering material—sand, coal, etc.—enters the frames *a* at the top in form of a thin paste and is transformed into a solid cake, while the liquid runs off through the corrugations of the plates *b* and *c*.

After the filter has so been formed in the frames *a* (Fig. 8) the plates *c* are replaced by plates *d*, while the frames *a* and the plates *b* remain in their place. Now the material to be filtered enters at the top at *d*, passes from right to left through the frames *a*, which are filled with sand, coal, etc., and the liquid is discharged perfectly clear through the cock of plate

b. These presses are built of wood and iron, with or without washing arrangement.

A filter press for experimental laboratory purposes is built by most filter press manufacturers.

The most prominent builders of filter presses in the United



FIGS. 9 AND 10.—CENTRIFUGAL BASKET AND DRUM.

States are: The Allbright-Nell Company, Bushnell Press Company, J. B. & J. M. Cornell Company, John Johnson Company (Fig. 4), Niles-Bement-Pond Company, William R. Perlin & Company (see this journal, Vol. V, p. 67), Platt Iron Works Company, Robert S. Redfield & Company, T. Shriver & Company (Figs. 1, 2 and 2a), D. R. Sperry & Company, Sweetland Filter Press Company.

Centrifugal Separators.

Another appliance for separating liquids from solids is the

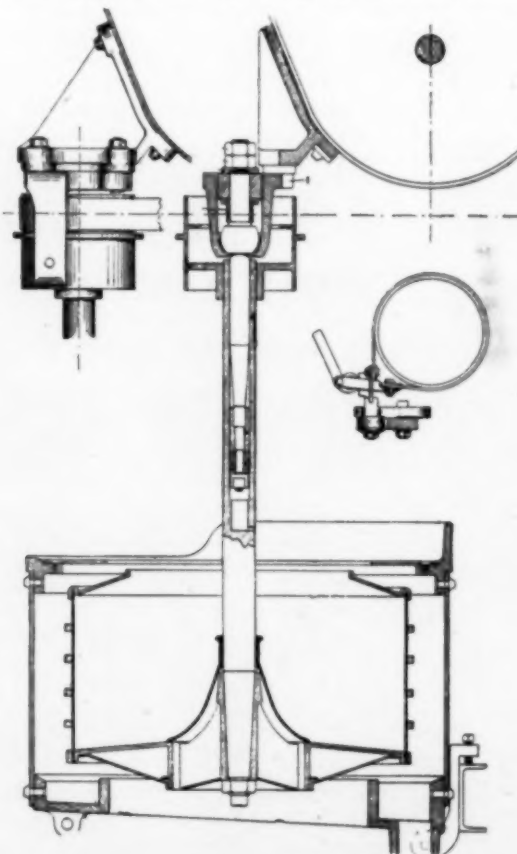


FIG. 11.—CENTRIFUGAL, WITH UPPER DRIVE.

centrifugal. The "basket" containing the liquid and solid substances is rapidly rotated, separation being effected by centrifugal force.

We distinguish centrifugals with upper drive and under

drive. The ones with upper drive require best workmanship and careful operation; though occupying little floor space they have the disadvantage that, on account of the shaft passing



FIG. 12.—BOTTOM-DRIVEN CENTRIFUGAL.

through, the interior of the basket is partly not accessible. This makes the discharging rather difficult. Other disadvantages are the necessity of using heavy foundations and the possibility of impurities getting into the material during lubrication.

Centrifugals with bottom drive occupy more space and have to be provided with a good regulating device; on the other hand they are easily accessible, and do not require a heavy foundation.

According to the material to be treated the interior of the

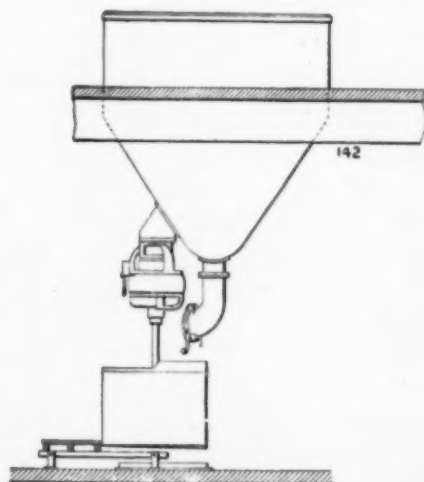


FIG. 13.—ELECTRICALLY-DRIVEN CENTRIFUGAL.

centrifugals is made of copper, steel, wrought iron, bronze, brass, nickel, aluminium, lead, clay or porcelain.

A centrifugal basket, as used by the American Tool & Machine Company, is shown in Fig. 9, the outside drum in Fig. 10. A belt-driven centrifugal (upper drive), built by the same company, is shown in Fig. 11. In Fig. 12 is shown a bottom-driven German centrifugal with movable basket and outside drum.

All belting, shafting and the throwing of oil is avoided by direct connection of centrifugals with electric motors. This method insures rapid acceleration, uniform speed, reduced floor space and extreme cleanliness. Fig. 13 shows (in outline) this type with upper drive as built by the D'Olier Engineering Com-

pany. This same firm also builds direct-mounted-motor under-driven centrifugals (Fig. 14).

It has to be borne in mind that the charge in the basket has to be uniformly distributed before starting the operation.

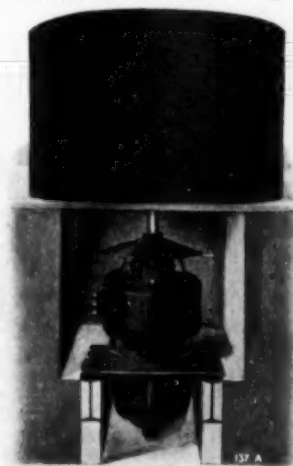


FIG. 14.—DIRECT-MOUNTED-MOTOR UNDERDRIVEN CENTRIFUGAL.

For laboratory or experimental use a 10-inch centrifugal is built by the American Tool & Machine Company.

Finally we want to mention the De Laval centrifugal clarifier which was designed originally for the separation of cream from milk, but is now used for various materials. It is built by the De Laval Separator Company.

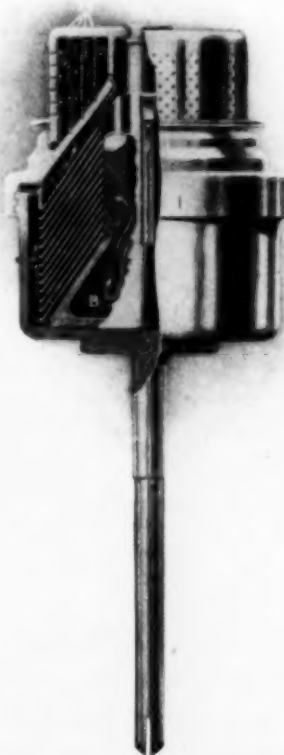


FIG. 15.—CENTRIFUGAL SEPARATOR.

Fig. 15 is a sectional view of that part of the De Laval machine in which the clarifying and filtering is actually accomplished. This is called the bowl, and consists of two separate compartments, which, since their duties are quite different, really perform the work of two independent machines.

The first or lower compartment is the clarifying chamber, the upper the filtering chamber. The clarifying chamber consists of a cylinder filled with a series of cone-shaped pressed-steel discs, one above the other, about one-sixteenth of an inch apart, and held rigidly in position by a central feeding device, called the "split-wing tubular shaft." The filtering chamber or upper compartment is also cylindrical in shape, but of less diameter than the clarifying chamber. Its side wall contains numerous perforations and is lined with the filter material "H," which may be paper, cloth, wool, etc., as governed by the requirements of the product handled. Within this chamber are placed three concentric cylinders dividing the chamber into four compartments, D, E, F, G. All three cylinders contain numerous perforations and are lined with filter material H. The

liquid first enters the bowl at the point A through the feeding device in the center and drops to the bottom of the clarifying chamber, and is then forced upward by the great centrifugal force and out through the wings of the tubular shaft into the spaces between the discs, which act as so many independent bowls.

In this operation from 90 to 95 per cent of the coarser dirt and sediment is removed and retained in the sediment pockets B and C. After leaving the discs the liquid continues to be forced upward out of the clarifying chamber into the compartment D of the filtering chamber. Here the same centrifugal

force drives the liquid through the walls of the three inner cylinders into compartments E, F and G, the filter material H, with which these cylinders are lined, catching and retaining all the remaining sediment and dirt not removed by the clarifying compartment.

Lastly the liquid is forced through the filter material lining H of the side wall of the filtering compartment itself, and is then delivered into such receptacle as may be waiting for it. This second operation, which consists of forcing the liquid through four independent linings or layers of filter material H, completely removes every last bit of cloud or flock.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

Electric Furnaces.

Electric-Furnace Reduction of Alumina by Means of Carbon.—

E. Viel, 883,594, March 31, 1908. Application filed Oct. 31, 1906.

To reduce alumina, silica, magnesia, chromite, etc., by means of carbon in the electric furnace, the following conditions are said to be necessary: Raising the temperature approximately to the "decomposition temperature," using the oxide and carbon (charcoal) in very finely pulverized condition and in exact proportion; for instance, according to the equation $Al_2O_3 +$

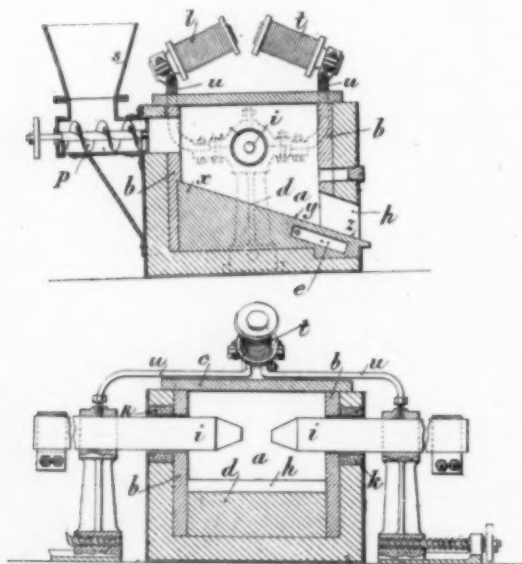


FIG. 1.—ALUMINIUM FURNACE.

$3C = Al_2 + 3CO$; removing the reduced metal (aluminium) immediately from the hot zone into a cold zone (the temperatures of these two zones in the case of aluminium being given as 3000° and 800° C. respectively); continuous operation so as to bring only small quantities of the mixture of oxide and carbon at a time into the excessively hot zone. A furnace for this purpose is shown in cross-section and longitudinal section in Fig. 1. The fusion chamber *a* is provided with carbon walls *b* and a carbon cover *c* and a sole *d* of a very steep inclination (25° to 30°). This sole is water-cooled at its lower end *e*, next to the tapping hole *h*. The powdered mixture is fed into the hopper *s* and introduced into the furnace by the screw conveyor *p*. The two electromagnets *l*, jointed on the legs *u*, exert a repelling effect upon the arc between the electrodes *i* so as to blow it down and bring it nearer to the sole *d*. By

adjusting the ampere turns of one or the other or both electromagnets it is possible to blow the arc into a predetermined zone of the sole (always between *x* and *y* in the upper diagram). The mixture of alumina and carbon drops from the screw-conveyor onto the hottest zone of the sole. Aluminium is reduced, flows off into the cold zone *yz*, while the carbon oxide escapes upwards. The reduction of silica in this furnace is said to be a more ready operation "because the volatilization temperature of silicon is 1650° , i. e., much higher than that of aluminium, and the difference between the volatilization temperature and the decomposition temperature (2500°) is much less. With this process it is also possible to treat silicates or analogous compounds in order to obtain compounds or alloys, such as silico-aluminium, ferro-chromium, etc. In order to obtain pure aluminium and silicon from silico-aluminium, it is necessary to heat the silico-aluminium to 700° upon the sole of a reverberatory furnace; the aluminium separates and the silicon remains as residue. The aluminium thus formed still contains 3 per cent of silicon; in order to obtain it pure, the silicon is burned like carbon in a Bessemer retort."

Manufacture of Silicospiegel.—E. F. Price, 882,418, March 17, 1908. Application filed Nov. 14, 1905.

The furnace is a vertical stack with downwardly-converging walls and a water-cooled, cast-steel plate at the bottom, forming one electrode. An iron ring at the top of the furnace forms the other electrode. Supported on this is an iron dome with a bell-and-hopper-charging mechanism. The furnace charge consists of a mixture of pyrolusite, silica, iron or iron ore and coke. Initial current paths may be provided between the electrodes to start the operation, but later on the charge itself acts as resistor. The zone of maximum heat is on the hearth in the bottom. The reduced manganese, silicon and iron form a molten alloy which collects on the hearth of the furnace and is withdrawn through a tap-hole. A layer of the alloy solidifies on the water-cooled base-plate and then serves as the lower electrode.

Ferrosilicon.—E. F. Price, 882,417, March 17, 1908. Application filed Nov. 14, 1905.

The furnace is of exactly the same construction as in the preceding patent. The charge is a mixture of silica, iron or iron ore and coke.

Manganese Silicide.—E. F. Price, 882,582, March 24, 1908. Application filed Nov. 14, 1905.

The furnace is of exactly the same construction as in the two preceding patents. The charge is a mixture of pyrolusite, silica and coke.

Ferro-Chrome Free from Slag.—J. B. Huffard, 882,637, March 24, 1908. Application filed July 10, 1907. (Assigned to Electro Metallurgical Co.)

When ferrochrome is made in the electric furnace, the slags

are highly basic and infusible, containing oxides of calcium, magnesium, chromium and aluminium in such proportions that the melting point of the slag is higher than that of the alloy. It is difficult to separate the slag from the alloy. In order to obtain a clean separation the inventor taps the ferrochrome and the slag from the furnace into a pot having a lining. The heavier alloy subsides and the superincumbent slag quickly solidifies or stiffens, while the metal beneath remains molten. The taphole in the bottom of the pot is then opened and the molten ferrochrome is run out into the ingot mold. The solid body of slag is then discharged from the pot, which is relined for future use.

Calcium Carbide Furnace.—E. Appleby, 882,733, March 24, 1908. Application filed March 26, 1906.

The furnace consists of a crucible formed of two vertically-disposed cylindrical receptacles telescoping with each other, the upper receptacle being open at the bottom. A pair of electrodes extend into the crucible at the lower end of the upper receptacle which serves as a hopper for delivering the material to the zone of activity between the electrodes. The lower receptacle is open at the top and closed at the bottom and forms a container for the completed carbide. The lower receptacle can be withdrawn downwardly away from the electrodes and the upper receptacle and can also be disconnected from its lifting mechanism, to be withdrawn toward one side and then tilted for discharging its contents.

Electric Furnace.—H. L. Hartenstein, 883,110, March 24, 1908. Application filed Nov. 30, 1906.

Mechanical details of construction of a gas-tight electric furnace with movable and adjustable electrodes. The hearth or melting pot is supported upon a truck so that it can be easily removed. When in use the upper edges of this pot fit closely with the lower edges of the downwardly converging walls of a closed upper compartment so as to make an air-tight joint.

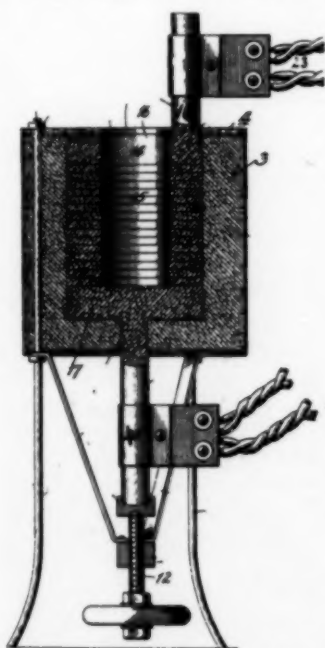


FIG. 2.—RESISTANCE FURNACE.

Within the upper compartment two electrodes are provided which can be raised and lowered at will by means of certain mechanism controlled from the outside.

Electric Furnace.—A. L. Marsh, 882,788, March 24, 1908. Application filed Sept. 19, 1907. (Assigned to Hoskins Co.)

3 in Fig. 2 is a casing of magnesite or fire-brick, with a cover 4 of asbestos-board. The furnace lining 14 forms at the same time the resistor and is made up of separate rings 15 of carbon in contact with each other, the top ring 16 and the bottom plate 17 being made of graphite and having terminal posts 18 and 19. The central opening is the heating chamber. By turning the screw 12 in one direction or the other, the register sections, which are compressed between it and the cover 4, will be loosened or tightened as to their surface contact with each other, so as to increase or decrease the resistance of the resistor. The rate of heat production and therefore the temperature may thus be easily controlled. (A description of the commercial type of this furnace will be found on another page in this issue.)

Induction Furnace.—K. A. F. Hiorth, 883,311, March 31, 1908. Application filed Nov. 20, 1906.

The object is to so arrange the primary inducting coil as to permit it to be used alternately or simultaneously for two or more furnaces. Claim 1 refers to "the combination with a plurality of furnaces, of a magnetic circuit common to all of them and a single magnetizing coil for said circuit." The arrangement adopted was already illustrated in our last issue, page 143, the first diagram of Fig. 2.

Electrolytic Furnaces.

Aluminium-Magnesium Alloys.—F. Von Kügelgen and G. O. Seward, 881,934, March 17, 1908. Application filed April 27, 1905. (Assigned to Virginia Laboratory Co.)

The electrolyte consists of magnesium oxide dissolved in magnesium-lithium fluoride (or a mixture of magnesium fluoride and the fluoride of a more electropositive metal). The cathode is molten aluminium. The magnesium, set free at the cathode, alloys with the aluminium. Magnesium oxide is fed continually into the electrolyte to replace that which is decomposed. Instead of dissolving simply magnesium oxide in the fluoride bath, both aluminium oxide and magnesium oxide may be dissolved in the bath and an alloy of aluminium and magnesium directly set free at the cathode. Instead of magnesium oxide magnesium oxychloride may be used.

Production of Aluminium.—H. S. Blackmore, 881,049, March 3, 1908. Application filed Oct. 29, 1904.

The electrolyte is a mixture of aluminium oxide and aluminium fluoride, to which calcium chloride may be added to increase the fluidity. The electrolyte is fused by the passage of the current. Soluble anodes are used, consisting of aluminium carbide (Al_4C_3 , aluminium methide). Aluminium is set free at the cathode, while oxygen and fluorine are liberated at the anode and immediately combine with the aluminium carbide to produce carbonic oxide (which escapes as gas) and aluminium fluoride which passes into the electrolyte. The aluminium carbide anodes must be replaced from time to time.

Electrolytic Processes.

Nickel and Copper from Ores and Matte.—E. Günther and R. Franke, 875,259, Dec. 31, 1907. Application filed June 18, 1907.

Garnierite and magnetic pyrites containing nickel and copper are smelted together so as to form matte or concentrated nickel or nickel-copper regulus. It is important to take care that the iron which may be contained in the ore is converted into slag as far as possible. The copper-nickel is subjected to electrolysis as anode in copper sulphate, containing free sulphuric acid; a copper sheet forms the cathode. At the anode iron, copper and nickel are dissolved, while the precious metals, sulphur, silicic acid, small portions of metallic sulphides, etc., form the anode slime. The latter is washed out, dried and treated with a suitable solvent for the sulphur. The residue is slightly roasted so as to obtain the copper, nickel, iron, etc., as oxides and the silver as sulphate. The residue is then treated with the spent electrolyte, whereby the free acid is neutralized and the oxides are converted into sulphates. The insoluble remaining portion, containing silver, gold, small portions of sulphides, sand, etc., is then treated to extract the precious metals. The sulphate-containing solution is treated with copper tailings to deposit the copper contained therein. The other metals are separated by fractional crystallization and chemical deposition. The last traces of copper are eliminated by nickel sponge. One obtains a technically pure sulphate of copper and nickel sulphate free from copper.

Since more copper is deposited on the cathode than is dissolved from the anode, it is necessary to add regularly new copper sulphate to the electrolyte. The latter becomes, of course, more and more rich in nickel. It is finally drawn off and partly used for treating the anode slime. From the rest the copper is extracted by evaporation, crystallization and subsequent electrolysis with insoluble anodes. The remaining liquid, which contains nickel sulphate, much sulphuric acid, a

small proportion of iron salts and other impurities, is neutralized with NiO; the impurities are chemically eliminated. A solution of pure sulphate of nickel is obtained, which is electrolyzed with insoluble anodes:

Nickel and Copper from Ores.—E. Günther and R. Franke, 879,633, Feb. 18, 1908. Application filed June 18, 1907.

Magnetic pyrites containing nickel and copper are converted into matte and the copper-nickel regulus produced is subjected to the action of chlorine either (1) in the form of gas or (2) in statu nascendi. (1) The ground copper-nickel regulus is introduced into a closed drum together with a solution of a chloride, such as sodium chloride, calcium chloride, etc., and treated with chlorine gas. If the temperature is regulated so as not to rise too much, the sulphides of the metals are converted into chlorides or subchlorides, while at the same time sulphur is set free. The solution is then filtered and freed from impurities and the resulting nickel-copper solution is electrolyzed with insoluble anodes. Copper is deposited at the cathode, while at the anode chlorine is developed. The electrolyte becomes poorer in copper during electrolysis. Therefore, fresh copper-nickel solution is added until the contents of the bath in nickel reaches a certain concentration. Then the copper is removed by electrolysis and its last traces by chemical precipitation. The resulting sub-chloride of nickel solution is electrolyzed yielding nickel and chlorine. (2) If the chlorine is used in statu nascendi, the copper-nickel regulus is used as anode in an electrolyte of a hydrochloric solution of copper chloride in mixture with an alkali or earthy alkali chloride, while a sheet of copper forms the cathode. Chlorine is evolved at the anode which converts the sulphides into chlorides liberating at the same time sulphur, and so on essentially in the same way as before.

Tin from Tin Scrap.—F. von Kügelgen and G. O. Seward, 883,139, 883,140 and 883,141, March 24, 1908. Application filed June 10, 1907.

According to 883,139 tin is recovered from tin scrap by a cyclic process, consisting of two steps. In the first step tin is removed from the scrap by means of anhydrous chlorine, yielding anhydrous stannic chloride. This is converted in the second step into an electrolyte by dissolving it in water and is electrolyzed, yielding tin and chlorine. The latter is dried and used to treat a second charge of scrap. The electrolysis is carried out, according to 883,140, in a diaphragm cell, but the electrolysis is not carried so far as to cause the stannic hydroxide to precipitate in the anode compartment (due to hydrolysis). Before this result occurs and when the current-density begins to notably drop, the electrolyte is discharged from both compartments of the cell, the anolyte containing stannic hydroxide and catholyte containing hydrochloric acid are mixed, and the mixture is concentrated by boiling it down to a density of about 1.4° to 1.5° Beaume, whereby the stannic hydroxide and hydrochloric acid are combined with the production of stannic chloride. The resultant solution is then returned to the electrolytic cell. A diaphragm cell is used, with an aqueous 20 per cent. solution of hydrochloric acid in the anode compartment (which may contain a small amount of stannic chloride) and a stannic chloride solution of specific gravity 1.2 in the cathode compartment. The cell is made of slate, the diaphragm of porous earthenware, the anodes of graphite and the cathodes of sheet tin. A cathodic current density of 4.2 amperes per square decimeter is used. As the electrolysis proceeds, the relatively high ionic velocity of the hydrogen cation causes a gradual transfer of hydrochloric acid from the anode to the cathode compartment, until finally the current-efficiency begins to notably drop. At this stage the catholyte may contain about 18 per cent. of hydrochloric acid and a little stannic chloride. Both solutions are thereupon removed from the cell and the old catholyte is transferred to the anode compartment, its hydrochloric acid content being brought up to 20 per cent. by the addition of acid. Stannic chloride is added to the old anolyte

until its specific gravity rises to 1.2 and it is then placed in the cathode compartment. The initial cell conditions are thereby restored and the electrolysis is resumed.

Pure Tin from Tin Dross or Waste.—A. J. M. Thiriot, 883,589, March 31, 1908. Application filed March 22, 1907.

The object is the electrolytic extraction of pure tin from stanniferous slimes, or from tin dross, or tin waste, or old tin plate, or from natural tin ores which have undergone an energetic oxidizing roasting and lixiviation. If the product treated is bronze, nearly all of the tin derived therefrom is found in the form of stannous hydrate, which it is advisable to preliminarily transform into stannic hydrate. This oxidation takes place after washing the slimes, being effected naturally, at the surrounding temperature, in the course of exposure of the slimes to the air, with a view to their desiccation, by atmospheric action. The process is carried out in three steps. The first step is the formation of a solution of stannate of sodium. The hydroxides of tin are treated at the boiling point with a 10 or 12 per cent solution of caustic soda in iron vessels; one liter of a 12 per cent caustic solution dissolves from 45 to 50 grams of tin. The mixture is maintained at the boiling point for about 1 to 1½ hours and is to be stirred during ebullition. Several vessels are employed to methodically enrich the liquor. Impurities entering into solution are antimony, arsenic, copper and lead. Antimony becomes antimonite of sodium, but in the course of electrolysis is changed into antimoniate, which precipitates. Arsenic forms arsenite, which becomes arseniate and accumulates in the liquor. The precious metals remain in the waste. The second operation consists in the purification of the stannate solution from copper and lead. The solution is heated to 70° C. and a concentrated solution of sodium sulphide is added whereby copper and lead are precipitated. The third step is the electrolysis of the purified solution, which is carried out in an iron tank with insoluble anodes of iron and cathodes of sheet tin or tin plate. The electrolyte is kept at a minimum temperature of 80°. The voltage is 2.4 and the current density 300 to 400 amperes per square meter of cathode, a single side only being counted. Twenty-five vats may be easily placed in series. The electrolyte is held sufficiently concentrated in stannate of sodium and is maintained in a fairly active circulation. The tin deposit is of a clear gray color and very adherent and coherent and is stated to contain less than 0.1 per cent impurities. If the current density is not too high, 0.8 gram of tin may be obtained per ampere-hour. (According to our Vol. V., p. 375, where this process was already described, the same is stated to be in commercial use in France.)

Electrodeposition of Iron.—S. O. Cowper-Coles, 884,075, April 7, 1908. Application filed Sept. 16, 1907.

Iron can be deposited in a form suitable for the production of tubes, sheets and wire, with a bright smooth surface resembling that of very highly polished iron, by maintaining the solution charged with iron oxide in suspension. Excellent results are obtained with a solution containing 20 per cent. of sulpho-cresylic acid saturated with iron, the current density being 100 amperes per square foot of cathode surface, the voltage 3.25 at the terminals of the iron electrodes, these being one-half inch apart and the temperature of the electrolyte 70° C. The temperature of the electrolyte considerably affects the quality of the iron. If it is much below 70° C. the iron becomes laminated and flakes off; if it is much above 70° C. the surface becomes covered with ridges or stream lines and cannot be used for commercial purposes without further treatment. "The sulpho-cresylic acid above mentioned is a cresol-sulphonic acid containing approximately 108 parts cresol and 98 parts sulphuric acid. The cresol contains ortho 35 per cent, meta 40 per cent. and para 25 per cent. This cresol is heated with sulphuric acid, yielding isomeric cresol-sulphonic acids." It is important that none of the oxide in suspension shall be deposited on the iron, otherwise it will be worthless for com-

mercial purposes. It is therefore advantageous, in producing sheets or tubes, to slowly revolve the cathode which may be arranged longitudinally or vertically. This also insures an equal thickness of deposit by changing the relative position of the anode and cathode. The iron produced from the sulphocresylic solution is exceedingly hard and when it is desired to produce soft tough iron, ferrous sulphate solution should be employed. It may be advantageous to add small quantities of carbon bisulphide from time to time to the electrolyte. "Iron articles produced as described above do not pit or corrode like iron which has been cast or wrought into the desired form, and this is probably due to the purity and uniformity of the metal." When requiring to produce steel articles "carbon is deposited with the iron" and after removal from the mandrel they are heated to a high temperature to convert the iron into a steel. In starting with iron carbonate or sulphide ore, the ore is roasted and mixed with crushed coke so as to form a filter bed and sulphuric acid solution is passed through the filter bed. The coke and iron oxide form an electric couple in the presence of the acid, thus facilitating the dissolution of the iron oxide. An external source of current may also be employed.

Recovery of Metals from Dilute Solutions.—S. B. Christy, 883,170, March 31, 1908. Application filed March 10, 1906.

The object is to recover gold, silver, copper from the very dilute solutions which result from the extraction of the metals from ores, whether these solutions be acid or alkaline, and whether the metals be combined as chlorides, sulphates, bromides, cyanides, etc. In order to recover the metals in a reasonable time, it is necessary to have numerous cathode surfaces of enormous area, and a similar number of anodes, but of smaller area, and to circulate the solution so as to bring it into intimate contact with anodes and cathodes in rapid alternation so as to maintain a constant supply of metal at the cathode for the electric current to precipitate. All the electrodes are made pervious and the solution is passed from one end of the cell through the series of electrodes to the other end of the cell. An arrangement similar to the series system for copper refining is employed; only the first anode and the last cathode being connected to the external circuit, while all intermediate electrodes ("compound electrodes") act as bipolar electrodes, one side being cathode, the other anode. To protect the anode side against the corrosive action of the electrolyte, it is made of artificial graphite, with perforations of an angle of 45°. The cathode side, which receives the metallic deposit, may be made of wire cloth or fragments of graphite, etc., held by cheese-cloth in close contact with the anode side. Other constructions of the "compound electrodes" are also described. For instance, "coke whiskers" (an occasional product of coke ovens) is very suitable for receiving the metal deposit. Common charcoal is also very suitable if made an electric conductor by heating it to a yellowish or white heat. It becomes a better conductor when charred under pressure and by the addition of small amounts of such substances as rosin, asphaltum, bitumen, etc. Charcoal in filament form made from "excelsior" is excellent.

Precipitating Gold.—John E. Greenawalt, 876,346, Jan. 14, 1908. Application filed April 3, 1905.

The object is to precipitate gold and silver from chloride solutions produced in the chlorination process. The precipitation is carried out at a temperature of 130° F. or more in a cell constructed as follows: In the lower part of the cell is placed a perforated floor on which sponge or shavings of lead, with from ½ to 1 per cent of zinc, are placed. The shavings are cut about one-sixteenth of an inch wide and a hundredth of an inch thick. In the upper part of the cell a porous jar is provided containing the carbon anode. The metallic shavings on the perforated bottom form the cathode. The solution is introduced into the cell near the bottom and just below the perforated floor. It then flows upwards and through the shavings. The cathode deposit is a black pulverulent slime consisting principally of lead, silver and gold, which is melted into bullion and the

metals separated. The recovered lead is melted and a small amount of zinc is added, after which it is cut again into shavings. With a depth of 6 inches of shavings a current of five amperes for every cubic foot of shavings is recommended.

Diaphragm Cell.—W. H. Rines, 878,425, Feb. 4, 1908. Application filed June 17, 1907.

This diaphragm cell, which is intended for sodium chloride electrolysis, is shown in Fig. 3; 12, 13 are the graphite anodes, while 2 is the perforated cathode having deep corrugations of V-shape. The object of the construction of the cathode is to cause the caustic formed to move by gravity from the sphere of electrolytic action so that it falls away freely from the cathode plate as rapidly as formed, thus preventing a dif-

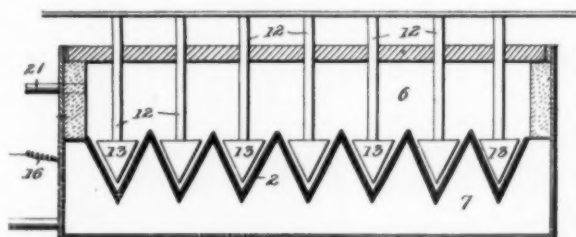


FIG. 3.—DIAPHRAGM CELL.

fusion into the anode chamber. The cathode is perforated and the top is coated with an insulating paint. A diaphragm conforming with the corrugations of the cathode, in form of a sheet of asbestos paper, is placed above the cathode plate. The sodium hydrate falls to the bottom of the tank. 6 is the anode compartment filled with sodium chloride, and 7 is the cathode compartment filled with water up to within about an inch of the cathode plate. "A liquid, inert to the products of electrolysis, is poured over the water and serves to prevent the carbon dioxide in the air from acting on or carbonating the caustic soda solution in the bottom of the cathode chamber and also has a balancing medium to retard diffusion."

Anode for Chloride Electrolysis.—M. Ruthenburg, 882,169, March 17, 1908. Application filed Sept. 20, 1901.

A chlorine-resisting electrode is "formed of a compound comprising silicon and another substance which is a conductor of electricity, such as metallic copper, carbon and the like, capable, *per se*, of resisting the action of chlorous acid." An alloy containing from 10 to 25 per cent of silicon is recommended.

Production of Chromic Acid.—M. LeBlanc, 883,651, March 31, 1908. Application filed Jan. 13, 1906. Assigned to Badische Anilin & Soda Fabrik.

The object is to produce chromic acid by anodic oxidation from chromium compounds of a lower degree of oxidation and particularly to regenerate chromic acid solutions which have been used for oxidizing purposes; for instance, for the production of anthraquinone from anthracene. A diaphragm cell is used, but the diaphragm does not extend to the bottom of the cell. The solution to be oxidized is introduced at the bottom of the cathode compartment. The electrodes are made of hard lead. The best results are obtained often only after a considerable period (for instance, 14 days) of undisturbed working.

Anode Support for Electroplating.—A. M. Hill, 882,110, March 17, 1908. Application filed April 15, 1907.

Mechanical details of a suspension arm for an electroplater's anode to enable an easy separation of the support from the anode and the convenient insertion of a new anode. The anode has a head formed with a flat transverse horizontal face which is at right angles to the longitudinal axis of the anode and is formed with side faces below the flat face. These side faces are inwardly inclined at an angle to the flat face and are likewise longitudinally inclined, so that the head partakes of the form of a double wedge shape. The supporting hook has a socket at its lower end which is adapted to engage

the wedge-shaped head of the anode. This socket contains a flat side which rests on the flat face of the head and also is formed with lateral flanges interiorly inclined in two directions so as to correspond to and engage with the double inclination of the sides of the head. The whole arrangement is so that an interlocking of the socket with the head may be effected when the two parts are connected together, the interlocking being of such a character as to prevent the hook from being disengaged from the anode except when moved in a particular direction for that purpose.

Electrotypes.—S. O. Cowper-Coles, 875,784, Jan. 7, 1908. Application filed March 5, 1906.

The interior of a circular vat is fitted with a framework of wood or other suitable material adapted to receive the trays or cases filled with wax forming the cathodes upon which the impressions are made for electrotyping. A frame is suspended in the center of the vat in such a manner that it can be rotated; it carries the copper anodes, which are perforated to permit of the free circulation of the electrolyte. The electrolyte is introduced through the bottom or center of the vat by means of a pump and is projected by centrifugal force into the molds on which the metal is to be deposited so as to remove any air bubbles that may be retained in the recesses thereof. In order to free the electrolyte from suspended matter, it is forced by the pump through a filter.

Electric Discharges Through Gases.

Nitric Acid from Air.—H. Pauling, 882,958, March 24, 1908. Application filed Aug. 12, 1902. Assigned to Westdeutsche Thomasphosphat-Werke.

The feature is the combination of an electric furnace with a regenerative gas-furnace. The furnace consists of a heating chamber in which the arc discharge takes place, and one or more pairs of heat regenerators surrounding the heating chamber. The air after having gone through the arc discharge, passes through the regenerators and is then subjected to steam to produce nitric acid. A fresh supply of air is then heated by the regenerators, etc.

Ozonizer.—A. C. Wood, 882,509 and 882,510, March 17, 1908. Applications filed Oct. 18, 1904.

Through a tubular electrode of copper, lined inside with nickel, extends a nickel-plated rod, coaxial with the tubular electrode, but insulated from it. The rod in the center is slotted longitudinally, so as to carry pointed combs from which a brush discharge passes off; or nickel brushes of circular or disk form are sleeved on to the rod so as to produce the brush discharge. The air to be ozonized is passed through the space between the tubular electrode and the rod in the center.

Batteries.

Storage Battery Plate.—W. Gardiner, 874,841, Dec. 24, 1907. Application filed April 26, 1906.

Claim 2 refers to "a storage-battery plate comprising a bar, a spacing-member mounted on said bar, a receptacle having a transversely slotted bottom and adapted to contain active material, and mounted on said spacing member, an absorbent sheet mounted between each of said receptacles and said spacing-members, and a frame cast integral with each of said members after the same have been assembled."

Storage Battery.—W. Gardiner, 877,889, Jan. 28, 1908. Application filed May 2, 1905.

The fifth claim refers to "a grid comprising oppositely disposed series of horizontal containing bars of a triangular cross-section set with apex outward, and a series of oppositely disposed vertical strengthening bars, each bar being provided with a series of inwardly flaring lugs or teeth, the teeth of oppositely disposed bars being in staggered relation to each other."

Storage Battery.—W. L. Merrin, 882,573, March 24, 1908. Application filed Oct. 5, 1896.

The containing vessel is made of corrugated sheet iron

which forms one electrode, while within the cell there are a series of corrugated iron sheets forming the other electrodes. The latter are so bent as to firmly hold and enclose the active material consisting of red lead. An alkaline electrolyte is used. In charging, lead is deposited on the negative electrode, while the active material of the positive electrode is oxidized to a higher oxide.

Storage Battery.—T. A. Edison, 880,978, March 3, 1908. Application filed Nov. 2, 1905. Assigned to Edison Storage Battery Co.

The first claim refers to "an electrode element, comprising a perforated inclosing pocket, non-deformable under normal working conditions, containing a highly compressed mass of active material and a network of conducting paths, formed of overlapping flakes, extending throughout the active mass and in contact with the pocket walls and with which the particles of active material are deformably compressed into intimate contact."

Storage Battery.—T. A. Edison, 880,979, March 3, 1908. Application filed Nov. 2, 1905. Assigned to Edison Storage Battery Co.

The first claim refers to "the method of making electrode elements which consists in introducing a mixture of particles of active material and flake-like conducting material within non-deformable inclosing pockets, and in applying a pressure to such material sufficient to crush or deform the active particles and cause them to substantially follow the contour of the conducting flakes."

Storage Battery.—T. A. Edison, 882,144, March 17, 1908. Application filed March 30, 1905.

Claim 25 refers to "a storage-battery electrode, comprising a sectional inclosing pocket having perforated walls, the sections of which are welded together, a multitude of conducting metallic flakes within the pockets, said flakes being welded together and to the pocket walls to form an integral conducting sponge-like or cellular structure and active material carried within the cells thereof."

Storage Battery.—J. W. Aylsworth, 880,957, March 3, 1908. Application filed April 28, 1905. Assigned to Edison Storage Battery Co.

The patent refers to the production of nickel hydroxide as used as active mass in the Edison battery. The feature of the invention is that hydrated peroxide of nickel, $\text{Ni}(\text{OH})_2$, can be formed in and plated out of a cyanide solution electrolytically at the anode.

RECENT METALLURGICAL PATENTS.

Iron and Steel.

Converter Process.—Alexandre Tropenas, the French metallurgist, best known through his modification of the Bessemer converter, patents a method of carrying out the converter process, suitable for comparatively small charges (880,253, Feb. 25, 1908). In order to produce sufficient heat to have the metal in fluid condition, he adds silicon, the heat of combustion of which yields the necessary temperature. This method presents the following difficulties: If silicon is added before the appearance of the carbon flame, it retards and prolongs the first period of the process which comprises the time from commencement of the blow to the time when the carbon flame appears; the result is that the lining is worn away and a bad operation is produced. If on the other hand silicon is added near the end of the operation at about the moment of the dropping of the carbon flame, the operator is deprived of the reliable signs furnished by the flame for the stoppage of the operation at the right moment so that it is exceedingly difficult to get the required rate of steel. Mr. Tropenas starts with a hematite pig iron containing as little silicon as possible. If such low silicious iron cannot be obtained, a mixture of pig iron and steel scrap may be charged in the cupola so as to

lower the percentage of silicon in the metal poured into the converter. On account of the low percentage of silicon the first period of the blow is very short. As soon as the second period begins, that is immediately after the appearance of the carbon flame at the mouth of the vessel, a small quantity of silicon (about 5 per cent), in the form of highly silicious pig iron or ferrosilicon, is added and the blowing is continued without stoppage of any sort until the flame drops. This drop is very characteristic and very clear. The steel contains no trace of the supplementary silicon which has been added and is not overoxidized. The final addition of ferro manganese is then made in the usual way and the steel is ready to pour into molds. It is stated that experience has shown that by this process with a converter of a half-ton capacity only it is possible to always obtain the same percentage of carbon and the same quality of metal without appreciable practical variation, and that the steel is sufficiently fluid to be carried in small ladles and to be poured into very small molds.

Modified Converter Process.—E. von Maltitz (879,480, Feb. 18, 1908) proposes to purify iron in a revolving vessel of egg form as shown in Fig. 1. The interior of the vessel 2 is provided with a refractory basic or neutral lining 26, and the trunnions are hollow to provide inlet and outlet passages 27 and 28. The blast pipe 29 terminates in a nozzle 31 which is so directed as

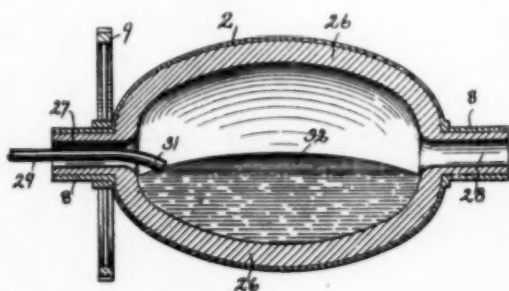


FIG. 1.—MODIFIED CONVERTER PROCESS.

to cause the blast to switch horizontally through the vessel across the surface and imping against that wall which rotates upwardly. After a charge of melted iron has been introduced, together with enough burnt lime to insure the production of a strongly basic slag, the apparatus is set in rotation (from 10 to 40 revolutions per minute for a vessel treating 15 tons of metal). The air-blast is then turned on, and on account of the direction of the nozzle noted above, the air blast impinges on the surface of the metal where it is freed from slag by centrifugal force. The blast traverses the length of the vessel and passes out through 28. The silicon and carbon are first oxidized with considerable evolution of heat; and with steadily rising temperature, a liberal quantity of the basic slag forms which eliminates the phosphorus. The process differs from the Bessemer converter, as the air is not blown through the metal, and it differs from the Tropenas converter by much more rapid operation on account of the rotation of the vessel. The rotation continually exposes fresh surfaces of metal from beneath the slag; further under the centrifugal action the impurities which are specifically lighter than pure iron are driven toward the axis of rotation of the vessels, viz., to the surface of the bath where their elimination takes place. The constant and thorough agitation also facilitates the elimination of occluded gases. When the slag contains enough iron oxides to act as a purifying agent, the blast is interrupted or restricted, in order to recover to a greater or less extent the iron in the slag. The slag may then be drawn off, and the metal recarburized and converted into steel, the rotation of the vessel insuring a perfect mixture.

Chilling Cast Iron.—Carborundum has been used in the past in the steel industry as a substitute for ferrosilicon, but a new and interesting application is proposed by Mr. R. C. Totten

(881,036, March 3, 1908). Chilled castings are generally made from charcoal pig iron on account of the quantity of combined carbon contained in it. Mr. Totten now produces chilling cast metal from pig iron made by the coke and coal process, with the aid of silicon carbide (carborundum). The coke or coal iron is melted in an air furnace and the proper proportion of silicon carbide is added to produce the proper depth of chill, carborundum continuing some 30 per cent. C.

Blast Furnace Slag for Cement Making.—F. W. Wood (883,770, April 7, 1908) proposes a modified and less expensive procedure in treating blast-furnace slag. To secure the crystalline state essential to the highest cementing qualities, the slag must be held for some minutes in a pasty condition. For this purpose the molten slag is delivered into an inclined revolving tube or barrel through which it passes from end to end. This tube is cooled artificially by water from the outside. The slag is thereby divided into pasty nodules, which tumble over and over in contact with one another. The size of the nodules is governed by the rate of rotation and the rate of cooling. By the gradual cooling a "molecular condition" is obtained which is claimed to be preferable for cement-making purposes to the vitreous condition produced by the sudden cooling employed in ordinary practice. In some cases it is advantageous to subject the mass, while cooling, to the action of a strong current of air. The slag, as it leaves the blast furnace, contains from 1½ to 3 per cent. of sulphur in form of calcium sulphide, which is highly objectionable in cement, since in contact with air it slowly oxidizes to calcium sulphate with resulting disintegration of the mortar. By the exposure of the highly heated slag to the oxidizing effect of an air draft, a considerable portion of the sulphurous contents is oxidized within the barrel with the formation of calcium sulphate, which is not objectionable in cement.

Gold and Silver.

Cyanide.—Low-grade, highly refractory sulphide ores are treated by the cyanide process in the following way, devised by J. E. Porter and A. L. Clark (880,821, March 3, 1908): The crushed ore, mixed with the cyanide solution, is placed in a tank having a false bottom of a porous material like earthenware. Porous cylinders of earthenware are inserted within the mass of the ore. Compressed air is then driven through the pores in the false bottom upward into the ore mass and also from the inside of the porous cylinders through the pores into the ore mass outside, so that the whole mass is constantly agitated by forcing through it very minute bubbles of air, which aids in the effect of the cyanide solution. The mass is simultaneously heated and an addition of an alkaline earth oxide is made to neutralize any acidity. After the metallic values have been extracted by the solution the air pressure is withdrawn from the cylinders and suction is now applied through the cylinders, thereby filtering the solution containing the metallic values into the porous cylinders and withdrawing them to an outside reservoir. At the same time compressed air is still forced through the false bottom in order to agitate the slimes and prevent them from adhering in muddy layers on the outside surfaces of the cylinders.

Chlorination.—G. Gurney, of Berkeley, Cal. (871,766, Nov. 19, 1907), proposes to extract gold and silver from sulphide ores by a treatment in a saturated solution of common salt containing 6 per cent. ferric chloride and 1 per cent. hydrochloric acid. The ferric chloride is reduced to ferrous chloride with simultaneous formation of gold and silver chlorides. The solution is then electrolyzed, precipitating gold and silver on the cathode and oxidizing the ferrous to ferric chloride at the anode.

Chlorination.—John E. Greenawalt, of Denver, Colo. (873,309, Dec. 10, 1907), endeavors to avoid metallic losses in chloridizing roasting. One-half to one per cent. of common salt is added to the ore in the roasting furnace so as to act upon the

baser elements in the ore. The red-hot ore is then introduced into a gas-tight chloridizing chamber, where it is subjected to a stream of chlorine gas which chloridizes the precious metals and converts them into soluble chlorides. The ore passes downward and through an artificially cooled zone, where most of the volatilized fumes are precipitated. The escaping gases and steam caused by moistening the ore in the screw conveyor (which leads from the bottom of the chloridizing chamber to the roasted-ore car) are passed into a condensing chamber, where the remaining values contained in the gases are completely precipitated.

Filter.—H. W. Blaisdell and H. A. Brooks, of Los Angeles, Cal. (875,687, Jan. 7, 1908), describe a filter leaf with passage-ways of sufficient capacity to permit rapid and uniform filtration



FIG. 2.—FILTER CONSTRUCTION.

over the entire surface; 1 and 2 in Fig. 2 are sheets of fabric sewed together at 3, where 4 are strips of foraminous material; 5 are distenders with grooves 6 and 7. At the lower end of the filter leaf a perforated pipe is provided into which the liquid runs from the grooves 6 and 7 and is then discharged.

Copper and Nickel.

Smelting Copper-Nickel Sulphide Ores.—J. T. Carrick and S. Pattison (882,234, March 17, 1908) treat pyritic copper or nickel or copper-nickel sulphide ores, which are poor in sulphur available as fuel, by the following method, the object of which is to economize the consumption of coke or other fuel. The matte such as is ordinarily produced is treated with dilute sulphuric acid for the production of sulphuretted hydrogen and this gas is used as fuel in the cupola during a subsequent blast. For instance, when pyritic copper-nickel ore is under treatment the matte produced by the smelting of a first charge is a mixture of the monosulphides of copper, nickel and iron. It is withdrawn from the cupola, pulverized and treated with dilute sulphuric acid in a digester. The sulphuretted hydrogen evolved from the digester is passed into a gass holder, from which it is passed by pipes to a special set of tuyères adjacent to the air tuyères of the cupola. In burning with the air the sulphuretted hydrogen supplies the additional heat necessary for the reduction of further quantities of ore. The intense heat produced in the neighborhood of the tuyères prevents the formation of crusts at that point and enables the whole series of tuyères to be kept continually in blast. The use of sulphuretted hydrogen has the further advantage that the percentage of sulphur dioxide in the flue gases is increased, so that it becomes possible to produce sulphuric acid from such gases when it might otherwise be impracticable. The acid thus produced is utilized for digesting a matte for production of a further quantity of sulphuretted hydrogen. From the residue in the digester the high-grade insoluble copper sulphide containing a small percentage of nickel sulphide is separated and may be shipped to special treatment works or treated upon the spot as may be convenient, while the liquor which may contain a percentage of copper is treated, for instance, by crystallization for the recovery of the iron and nickel sulphates or electrolytically for the recovery of the copper and nickel.

Nickel and Copper.—Concerning the production of nickel and copper from garnierite and magnetic pyrites containing nickel and copper, see the electrolytic process of E. Günther and R. Franke, described in the Analysis of Current Electrochemical Patents in this issue.

Copper from Roasted Pyrites.—A. Tixier and C. Tortel (875,012, Dec. 31, 1907) propose to extract copper, etc., from iron pyrites, as follows: The iron pyrites are roasted and mixed in a dry state with 2 to 5 per cent. of commercial chloride of lime. The mixed ore is then introduced into vats, where it is sprayed with dilute sulphuric or hydrochloric acid and heated to a temperature of 70 to 80° C. The reaction is allowed to proceed for 12 to 24 hours, whereby copper, lead, zinc, silver, etc., are dissolved. The liquors are drawn off and the residue is washed with lukewarm dilute acid and then with water. The iron residues are practically free from foreign metals.

Zinc.

Continuous Zinc Furnace.—The disadvantages of our methods of zinc smelting are well understood—small units, intermittent operation, large losses of heat and high labor charge. It is very interesting to note how such an experienced zinc metallurgist as Dr. Franz Meyer endeavors to overcome these difficulties (879,482 and 879,483, Feb. 18, assigned to Metallurgical Company of America). His furnace is shown in Fig. 3. The two upper diagrams are vertical cross-sections; the one on the left hand is a section on the line 2-2 of that on the right

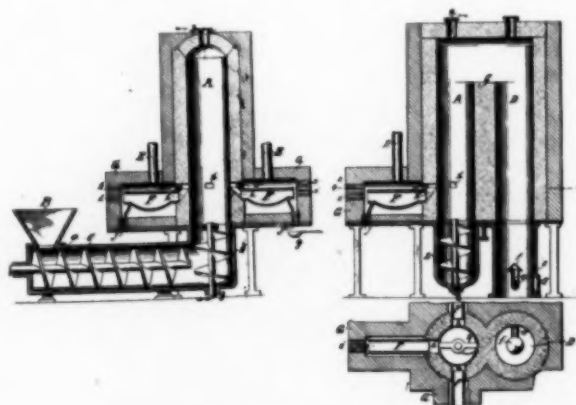


FIG. 3.—ZINC FURNACE.

hand, while the right-hand diagram is a section on the line 3-3 of that on the left hand. The lower diagram on the right is a horizontal section on the line 4-4. Roasted zinc blende or other zinc ore in form of oxide is mixed with coke or anthracite coal in sufficient quantity for reduction of the oxide and for the generation of the required heat. The mixture is fed through the hopper *B* into the conveyor *C*, which feeds it into the bottom of the stack *A*. It is then moved upward by means of the screw conveyor *b* within this stack and in this way the charge is made to rise gradually up to the top of the stack *A*. In rising upward the charge meets the hot products of combustion and of reaction which travel downward in the opposite direction, and the coke and anthracite coal of the charge becomes ignited. As soon as the charge attains the necessary temperature the zinc oxide is reduced and the zinc is vaporized. These vapors are driven down together with the products of combustion and leave the stack through the apertures *h* into the chambers *G* containing the condensers *F*. The zinc condenses within *F* and is periodically withdrawn from *a*, while the uncondensed waste gases occupy the spaces in *G* around *F* and maintain *F* at a temperature sufficient to keep the zinc in molten state. The non-condensed products of reaction finally leave through the exhaust pipes or chimneys *E*. The solid charge in ascending the stack *A* upward is subjected to a steadily increasing temperature, and the maximum temperature is reached at the top *C* so that practically all the zinc is driven out of the ore. The ash of cinder passes over *C* into the ash stack *D*. Here the hot ash gives off its heat to the cold air admitted through the inlet pipes *e*. The movement of the air upward through *D* and downward through *A*

is produced partly by devices forcing the air or gas through the admission pipe *E* and partly by the draft induced by the chimney stacks *E*. The air entering through *e* into *D* while ascending is heated by the ash and at the top *C* meets the carbon of the furnace charge. This is burned to CO_2 , which is reduced by the deep bed of glowing carbon below to carbon monoxide. The carbon monoxide gas escaping through *E* may be utilized for heating or power purposes. To increase the reducing effect, either CO or CO_2 , or both, may be admitted into the charge either at the top of the main stack or through the pipe *s*. Further solid or liquid fuel may be introduced through the inlet pipes *m* at the top. Several modifications of the apparatus are described in patent 879,483.

Zinc-Retort Residues.—Considerable quantities of zinc as well as of fuel are contained in zinc-retort residues. G. Stolzenwald (881,355, March 10, 1908) proposes to utilize them in connection with materials containing zinc, such as have already been used for making zinc-white, in which case the materials have been mixed with fuel and limestone. Stolzenwald mixes the zinc-retort residues with materials containing zinc without any other addition or flux, and heats the mixture in a long bedded continuously acting furnace. A suitable proportion of the mixture would be 100 parts of material containing 10 per cent. zinc with 20 parts of zinc residue containing 2 per cent. zinc and 22 per cent. carbon. During the gradual advance of the substances into the hotter part of the furnace there occurs a reduction of the oxide of zinc with simultaneous develop-

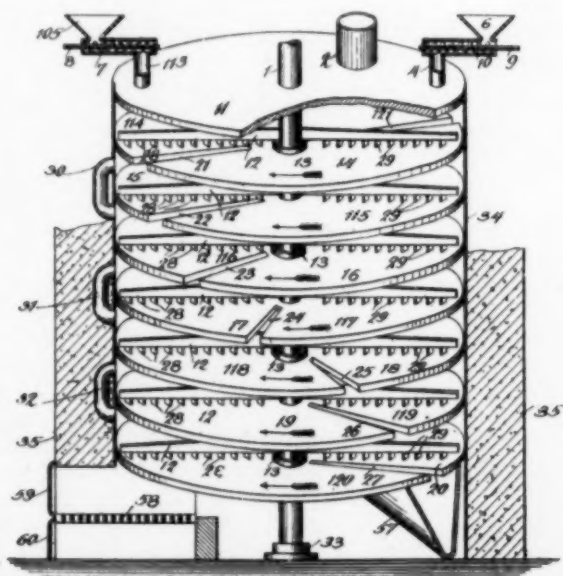


FIG. 4.—ROASTING FURNACE.

ment of zinc fumes, which latter oxidize then again into zinc oxide, when they arrive in the rear part of the furnace. The inventor thinks that if this process is used it does not matter much in zinc smelting practice whether somewhat more or less zinc remains over in the residue, since it can later be recovered from the residue.

Roasting Furnace.—Walter G. Swart (879,842, Feb. 18, and 882,217, March 17, 1908) patents the roasting furnace shown in Fig. 4. A series of hearths are mounted revolvable within the casing 35, each hearth being divided by diametrical slots, such as 21 and 121, into two halves. The non-aligned slots 21, 22, 23, etc., form a plurality of hearths on each level. The hearths rotate in the direction of the arrows. (The grate 58 below the hearths is fired through the doors 59 and 60 and may be used to send the gases of combustion upward through the slots of the hearths and also through the circulating pipes 30, 31, 32, which connect the spaces between successive hearths.) The

sulphurous gases pass off through the flue 2 to be utilized for manufacture of sulphuric acid. The principal feature of the process is that two different materials are roasted simultaneously, but out of contact with each other in the same furnace, namely, zinc blende (or galena in case of lead), which is fed through hopper 6 and conveyor 10, and iron pyrites, chalcopyrite, pyrrhotite, etc., which is fed through hopper 105 and conveyor 7. The object is two-fold, first to roast the ore without any coal, the whole heat being produced from the oxidation of sulphur, and, second, to produce richer sulphurous gases better adapted for the manufacture of sulphuric acid.

Aluminium.

Aluminium-Magnesium Alloys.—Certain aluminium and magnesium alloys have many valuable properties. They are easily cast, are lighter than aluminium and are easily machined. Their melting point is low and it is possible to make castings at a dull red heat. These alloys might supersede pure aluminium for certain purposes were it not for the high cost of magnesium and for the loss of magnesium when the metals are melted together to form the alloy. Concerning a new process of F. von Kugelgen and G. O. Seward for producing these alloys, see abstract in the Analysis of Current Electrochemical Patents in this issue.

Treatment of Fines.

Desulphurization and Sintering of Sulphides.—Sulphide fines have been treated in the past by mixing them with fines, igniting the mixture at the bottom and blowing a blast of air upward through the mixture. This is effective as far as desulphurization is concerned, but only the lower portions are sintered together, while the upper portions, on account of the agitation, remain pulverulent as fines. Arthur S. Dwight and R. L. Lloyd, of Cananea, Mexico (882,517, March 17, 1908), reverse this method by firing the mixture at the top and blowing the air downward through the mixture. The result is that both the desulphurizing and the sintering are complete and uniform throughout the entire charge and practically simultaneous. The method is applicable to galena or iron and copper pyrites. The same inventors (882,518, March 17, 1908) render this process continuous so as to permit a rapid treatment of fines with the production of a relatively thin cake or biscuit of sinter. The ore mass is moved itself bodily relatively to the igniter which is stationary. For this purpose the ore mass is supported on a pervious screen and the screen and the ore mass are together advanced while the mass is ignited at the top at a place between the point of feed and the point of discharge, and an air blast is passed downward through the mass toward the screen.

Converter Process.—Ores and mattes mixed with sulphur, coke and flux have been blown in the past in a converter in order to sinter the mass preparatory to further metallurgical treatment and as a substitute for the more expensive briquetting. J. Savelsberg, of Papenburg, Germany (875,852, Jan. 7, 1908), remarks that such a sintering process is applicable to such ores which do not need a special addition of sulphur, as they contain sufficient sulphur to form a matte. This process is also applicable to such ores as iron ores, which for melting do not need sulphur at all. In this case it is sufficient to add to the mixture finely divided coke to furnish the necessary heat. The finely divided ore mixture thus obtained is blown in a Bessemer converter until sintered. This sintered mass is broken up and fed direct to the blast furnace. The fluxes used are acid or basic fluxes, such as quartz rock, limestone, magnesite and the like, a protecting lining in the converter not being necessary for carrying out the process.

Treatment of Smelter Fumes.

In our last issue (page 136) we commented editorially on the problem of fumes from chemical works and smelters. For very good reasons numerous metallurgical engineers are en-

gaged at the present in finding a solution of the problems involved, as is indicated by a series of patents recently issued in this field. The object is generally twofold—first, to get rid of those constituents of smelter fumes which are deleterious to animal and vegetable life and which constitute legally a "nuisance," and, second, to recover any metallic values.

Absorption of Fumes in Water.—Most of the inventors treat the fumes with water and their arrangements differ only in the way in which the fumes are commingled with the water. J. T. Yates and J. Devey, of Lehi, Utah, and W. B. Richan and W. A. Devey, of American Fork, Utah (879,023, Feb. 11, 1908), use an apparatus consisting of an elevated scrubber tank, closed at top and bottom, and of an overflow settling tank. The two are connected by a pipe leading from the top of the scrubber to the bottom of the overflow tank. Both tanks are filled with water and the smelter fumes are introduced through a pipe with perforated distributing coil in the form of minute jets into the water in the scrubber tank. They pass upward through a series of screens and are thrashed by revolving beaters in the top of the tank so as to promote their intimate contact with the water and the precipitation of the solid constituents. The gases then pass over to the bottom of the overflow tank and pass through the same in a zigzag course, being again thrashed by means of beaters.

G. Morby, of Robinson, Utah (880,747, March 3, 1908), introduces the fumes at one end of a horizontal inclined flue, water flowing through the flue in the opposite direction. While the fumes pass through the flue they are met by showers of water from above and by air blasts. The air blasts tend to promote a more perfect combustion and the water spray tends to absorb and condense the sulphurous and other gases. The condensation products are taken up and carried off by the water on the bottom.

S. I. Clawson, of Salt Lake City, Utah (880,506, March 3, 1908), uses a revolving wheel in water. The fumes are introduced through a hollow central pipe into the wheel and are discharged through perforations in the periphery of the wheel. Fans are provided at the top to further agitate the water so as to thoroughly mix the fumes with the water.

John R. Moffitt, of Denver, Col. (882,073, March 17), passes the fumes from the chamber where they are volatilized to a hood located at the extremity of the conduit, the hood being immersed in water. The top of the hood is perforated, the walls being formed by punching holes inwardly, producing a jagged inner surface. As the fumes enter the hood they are compelled to travel along this jagged surface whereby they are broken up so as to cause a thorough and even distribution of the volatile products through the liquid, thus resulting in thorough condensation of all condensable elements.

Treatment with Carbon.—G. C. Carson, of Denver, Col. (876,437, January 14, 1908), endeavors to reduce the oxides in the fumes by means of carbon in the apparatus shown in Fig. 5. The fumes enter the chamber through *a* and are forced downward through the column of cold carbon until they enter the incandescent zone at *b*, which has been created by forcing an air blast into kindled carbon through the tuyer *c*. Here the oxides are reduced, like $\text{CO}_2 + \text{C} = 2\text{CO}$, $\text{PbO} + \text{C} = \text{Pb} + \text{CO}$, $\text{ZnO} + \text{C} = \text{Zn} + \text{CO}$. All the elements reduced from the gases which fall into the ash of the consumed carbon are washed through the grate bars with the ash, by giving the water which acts as a seal a jig motion, and are collected at the bottom of the pit beneath the apparatus. Part of the elements will be heated so high as to be in form of vapor and will ascend as such upward into the column on the right where they are cooled and allowed to pass off through the niches into the condensing chamber *x*. The carbon monoxide and nitrogen continue on up through the cold carbon and into the pipe to the gas mains where they are available for heating or lighting. The escaping fumes are quite cold so that there is no loss of heat.

Treatment with Comminuted Slag or Waste Ore.—S. S. Sørensen and G. C. Westby, of Murray, Utah (875,222, December 13, 1907), bring the smokes into direct and intimate contact with finely powdered slag, slag wool, tailings or finely divided waste ore, mixed with water, whereby the sulphur gases in the smoke combine with the bases and metals in these products, forming sulphates, thionous and thionic salts, etc. The metallic values are then precipitated from the solution. If the latter contains gelatinous silica it is necessary to prevent its precipitation if the solution be not sufficiently acid, and sulphuric acid is added to the solution which is then heated and boiled. The copper, etc., are thrown down as sulphides by the action of the reducing salts present in the liquor. When it is desired to precipitate the last traces of the sulphureted metals, a solution of FeS , Fe_2O_3 and FeSO_4 is added which is obtained by treating metallic iron scrap by a solution of sulphurous acid or crude scrubbing liquors containing free SO_2 .

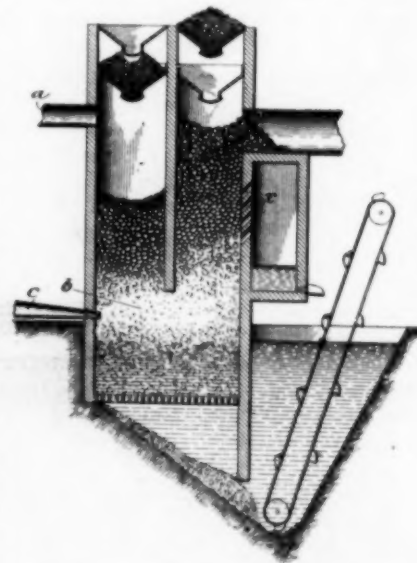


FIG. 5.—TREATMENT OF SMELTER FUMES.

Where commercial conditions make it desirable, the sulphuric acid liquor used for addition to the solution is made up as follows: A solution of ferric sulphate or an oxidized scrubbing liquor is added to pyrites, then a little sulphurous acid or smoke scrubbing liquor is added to the mixture and the whole is heated, the reaction being $\text{FeS}_2 + 7\text{Fe}(\text{SO}_4)_3 + 8\text{H}_2\text{O} = 15\text{FeSO}_4 + 8\text{H}_2\text{SO}_4$. Various modifications of the treatment of the liquor, according to local conditions, are described.

SYNOPSIS OF PERIODICAL LITERATURE.

Electrochemistry.

Production of Earth Alkali Metals.—In *Zeit. f. Elektrochemie*, March 20, F. von Kùgelgen gives some notes on his and G. O. Seward's method of producing metallic calcium by electrolysis. This has already been described in our columns (our Vol. V, p. 414). It is emphasized that this method is very essentially different from that of Borchers and Stockem.

Density of Graphite.—H. Le Chatelier and S. Wologdine (*Revue d. Metallurgie*, March) have made a series of determinations of the density of graphite and find that all natural or artificial graphites when completely purified have an identical density of 2.255 at a temperature of 15° C. in relation to water at 4°.

Commercial Silicon.—F. Limmer states in *Chemiker Zeit.*, Jan. 11, that high-grade crystalline and fused silicon is commercially available, but that commercial amorphous silicon is generally very impure and has especially a very high percentage of silica. All the samples of amorphous silicon which he had in his possession contained at least 20 per cent. of silica, one as much as 66 per cent.

Electroanalysis.—In *Zeit. f. Elektrochemie*, Feb. 21, F. Foerster replies to the criticisms of Classen and Fischer (our

March issue, p. 124). The same journal of March 20 contains a reply by A. Classen and a note by F. M. Perkin.

Nickel Analysis.—A. Fischer describes in *Chemiker Zeit.*, Feb. 22, a method for rapid separation of nickel and zinc, which is said to be specially suitable for the analysis of nickel alloys, like nickel coins, etc. The main conditions are: two concentric cylindric screen electrodes; composition of electrolyte 5 grams ammonium sulphate, 1 to 3 grams sodium sulphate, 30 cc ammonia (0.91), 0.15 gram each of nickel and zinc as sulphates, total volume 250 to 300 cc; the cathode potential to be controlled according to the compensation method; potential difference of the mercurous sulphate—2n—electrode against the cathode = 1.355 volts; temperature, 90 to 92° C., time of separation, 20 minutes.

The electrolytic separation of nickel and zinc by the Hollar-Bertiaux method is discussed by F. Foerster and W. Treadwell, Jr., in *Zeit. f. Elektrochemie*, Feb. 21. The separation is satisfactory at or above 90° C. in an ammoniacal solution, if a sulphite is added. However, the determination of the nickel itself is then inaccurate, since the nickel deposit contains sulphur. To avoid this error the nickel is to be redissolved and to be redeposited at ordinary temperature in absence of sulphite; or, for approximate determinations, it may be sufficient to apply a certain percentage correction.

Silver Voltameter.—In using the silver voltameter, a platinum crucible is generally used as cathode, but the percentage of accuracy of this arrangement is very low when very minute quantities of silver only (of the order of a milligram) are deposited. E. Bose and F. Conrad (*Zeit. f. Elektrochemie*, Feb. 21) propose to use a very thin platinum wire as cathode and to determine the deposited silver by means of Nernst's microbalance. The authors believe that most irregularities of the silver voltameter may be explained on the hypothesis of the presence of half valent silver.

Electrode Potentials.—In *Zeit. f. Elektrochemie*, March 6, Wl. Kistiakowsky discusses the methods of measuring electrode potentials against a calomel standard electrode. The primary object of his investigation was to measure the potential of the magnesium electrode. Among the more important precautions which are necessary in such measurements, according to the author are the following: At the point where the metallic electrodes touch the surface of the electrolyte, they must be protected from the action of atmospheric oxygen. Further, the arrangement of the experiment must enable one to rotate the electrode, on account of the necessity of removing local concentration differences. How far this is accomplished can only be determined by comparison of the e.m.f.'s with electrodes at rest and in rotation and after rotation. In many cases it is necessary to eliminate the effect of the varying oxygen concentration in the electrolyte; in general, it seems that the influence of the oxygen on the e.m.f. is the greater the more noble the metal of the electrode. The surface of the electrode must be polished; when dry and wet it must be microscopically examined.

Conductivity and Temperature.—In *Zeit. f. Elektrochemie*, Jan. 31, E. Rasch and F. W. Hinrichsen develop, on the basis of a formula of van't Hoff, the following relation between electric resistivity and temperature: $T \log W = C T + n$, where T the absolute temperature, W the specific resistance and C and n constants. This relation is shown to hold good for dielectric substances (insulators), figures being given for antimony trichloride, glass, china, oils, water and ice (where there is a sudden break in the curve at the melting point), and an arc electrode made from zirconium oxide and yttria.

Ionic Mobility and Its Temperature Coefficient.—In *Zeit. f. Elektrochemie*, Jan. 31, E. Rasch and F. W. Hinrichsen show that for many simple ions in aqueous solution the following relations exists between ionic mobility l and its temperature coefficient a , both at 18° C., $a \log l = \text{const}$. In *Zeit. f. Elektrochemie*, March 13, F. Kohlrausch remarks that he has proven

before this relation, but that, to be valid in general, another constant must be added. The relation holds good for ions of univalent elements, and may be written in the form $a(C + \log l) = A$, where A and C are constants, or $(a - P)(R + l) = Q$, where P, Q, R are constants.

Iron Salts.—E. Müller and F. Kapeller discuss in *Zeit. f. Elektrochemie*, Feb. 14, the reducing and oxidizing action of iron salts. Ferrous salts are reducing agents, ferric salts oxidizing agents. But their reducing or oxidizing effect is not always the same. Thus it is shown that the reducing action of ferrous salt solutions may be greatly enhanced by fluorine ions.

Anodic Polarization.—In *Zeit. f. Elektrochemie*, March 6, H. W. H. Schellhaass discusses the abnormal anodic polarization due to nitric acid.

Aluminium Carbide.—C. Matignon describes in *Chemiker Zeit.*, Jan. 15, various methods of preparing aluminium carbide C_3Al without the use of an electric furnace.

Chemical Apparatus.

Porous Materials as a Substitute for Cocks in Vacuum Apparatus.—In vacuum apparatus cocks represent a weak point since there is always a liability of leakage. A. Stock (*Chemiker Zeit.*, Jan. 8) refers to a very simple and practical device which has been proposed by K. Prytz. He suggests to



FIG. 1.—
POROUS
PLUG FOR
VACUUM
VESSELS.

use no cocks at all, but to substitute therefor porous plates which are permeable for air, but not for mercury. The principle is indicated in Fig. 1. A glass tube A, closed at its bottom, has at its top a stopper of porous material. Above the stopper some mercury is placed so that there is no connection between the gas inside the tube and the air outside. If now a second smaller glass tube, B, which is provided at its lower end with an analogous porous stopper, is so inserted into the top of the first tube that the two stoppers make contact with each other, the atmosphere within A and the atmosphere within B communicate with each other through the two stoppers. If B is connected to an air pump it is possible to produce a vacuum in A. By simply removing the tube B the connection of A with the air pump is broken, and there is no possibility of air entering into A. It is, of course, now possible in the same way to introduce any other gas into A. Prytz used a stopper of firebrick. The present author has found that a mixture of clay with water glass and rubber may be employed for the manufacture of such porous material of great uniformity with respect to the size of the pores. This material is not attacked by dilute acids or by boiling water, and it has the great advantage that without any binding material it can be fused to glass. The porosity is comparatively great. With a difference in pressure of 60 centimeters mercury, 600 to 800 cc gas pass through a plate of 8 mm diameter and 2 or 3 mm thickness per minute. Mercury does not pass through the plate even if the difference of pressure is more than an atmosphere. For short experiments a porous valve may be made in a very simple way by pasting to the flat end of the tube a disc of hardened filter paper.

Sulphuric Acid.

Manufacture of Sulphuric Acid.—G. Oddo in *Chemiker Zeit.*, Feb. 12, refers to the crisis of the sulphur industry in Sicily, and proposes to utilize the Sicilian sulphur deposits for making sulphuric acid. He has made experiments in two Italian sulphuric acid plants, one equipped with Maletra furnaces and lead chambers, the other with Herreshoff furnaces and the contact process. The plant was equipped for the treatment of pyrites and very slight changes were required to use Sicilian ore in the same. He obtained from each ton of ore (31.75 per cent. S) 1 1/3 tons of chamber acid, which is almost the theoretical output (about 1 1/2 tons). In the cham-

bers with the Maletta furnaces only 1.75 per cent., and in the Herreshoff plant only 0.75 per cent. were lost in form of calcium sulphide or calcium sulphate. Besides the almost complete recovery of all the sulphur, the method has other advantages. The sulphur is completely oxidized on one hearth. The plant may be greatly simplified for the contact process, since (in contradistinction with the use of pyrites) an almost pure and higher concentrated gas is obtained. The acid produced is free from arsenic and other impurities. The residue may be directly used as fertilizer or for making paving stone or for the manufacture of cement. He thinks this method has an industrial future and describes some simple furnaces for practical work.

Iron and Steel.

Silicon as Reducing Agent for Manufacture of Low-Carbon Ferro-Alloys.—In *Stahl und Eisen* of March 11, B. Neumann describes some electric-furnace experiments for making ferro-alloys with silicon as reducing agent in order to reduce the percentage of carbon in the alloy. A furnace of the Heroult type is used with two carbon electrodes suspended in the slag, which is a mixture of 150 to 100 Al_2O_3 and 100 CaO . This mixture is first introduced and fused by turning on the current. The mixture of silicon and the oxide to be reduced is then introduced and the result of the reaction is the reduction of the metal and development of SiO_2 , which is absorbed by the alumina-lime slag. The reduced metal collects on the hearth of the crucible. In the first experiments a chrome-iron ore containing 31 per cent Cr and 13 per cent Fe was reduced by means of silicon in a furnace lined with chromite brick. The silicon used was a commercial high-percentage ferro-silicon containing 91.65 per cent Si and 1.03 per cent C. The result of the process was ferro-chrome containing 38.05 chromium, 3.09 per cent silicon and 1.56 per cent C. The high percentage of carbon is perhaps due to contamination of the metal from the carbon electrodes. The percentage of silicon could have been reduced somewhat by increasing the amount of slag used, but in any case some silicon (about 2 per cent) always passes into the metal or alloy, and this is the weak point of the process. Other experiments were made with chromium, titanium, tungsten and molybdenum. If pure silicon or ferro-silicon low in carbon is used, the alloy or metal produced is also low in carbon, but some silicon always enters into the metal. This is not thought to be disadvantageous for many purposes of the steel industry. The direct reduction of sulphides by means of silicon is stated not to take place according to the simplest formula, while the reduction by means of silicon carbide is said to have no advantage in comparison of reduction by means of carbon in the electric furnace.

Electrometallurgy of Iron and Steel.—A systematic summary of the different electric iron and steel furnaces proposed and in use is given by C. Le Chatelier in *Revue de Metallurgie*, March.

Gold and Silver.

Slimes, Sizing and Classification.—In a paper on the handling of slimes, by Edwin A. Sperry, in *West. Chem. and Met.*, of March, it is pointed out that the success of the final treatment depends very largely on proper sizing, classification and dewatering. Within the past two years, the question of screen-sizing down to even 100 mesh has been so thoroughly met that there need be no hesitation on the part of the designer to adopt one of the several screens made for this purpose. There are two principal types. One is the shaking or impact screen, which has been very successful, though, Mr. Sperry thinks, the necessity of numerous wearing parts militates against it; examples of this type are the Impact, the Centripact, the Imperial and the Wild screens. The second type is the washing screen, examples being the Callow and the King screens, "either of which has much to recommend it, both in efficiency and life, as well as cheapness of maintenance." It is an unfortunate fact that in ordinary mill practice in the endeavor to properly size the pulp for treatment, the use of the hydraulic classifier is

relied on to an extent beyond its warrant. But while hydraulic classification is theoretically imperfect, yet below a certain point, say, 100 mesh and finer, sizing by screen is certainly impractical, if not impossible, and hydraulic classification must be taken up. As to "cone sizers," two serious disadvantages are pointed out, one being the extra demand on the mill supply of

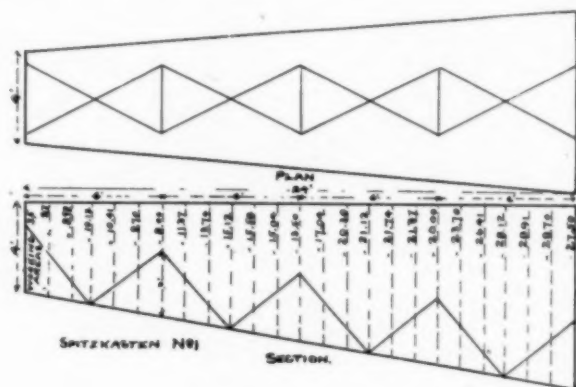


FIG. 2.—SPITZKASTEN.

water, and the other the excessive dilution of the pulp, which offers great difficulties in final treatment. The author considers the spitzkasten to be "one of the greatest assistants which the millman has at his command," and it is a constant source of wonder to him that they are not more universally employed. However, one of the most important fundamental principles is the absolute necessity of a thoroughly placid method of coaxing instead of a violent method of attempting to force the particles to do the will of the operator. In all operations connected with the handling and treatment of slimes the utmost importance must be given to the avoidance of currents or agitation. This applies to all devices, as well as to the so-called improvements to otherwise efficient appliances in which or by which any kind of agitation, constriction of flow, disturbances or currents of

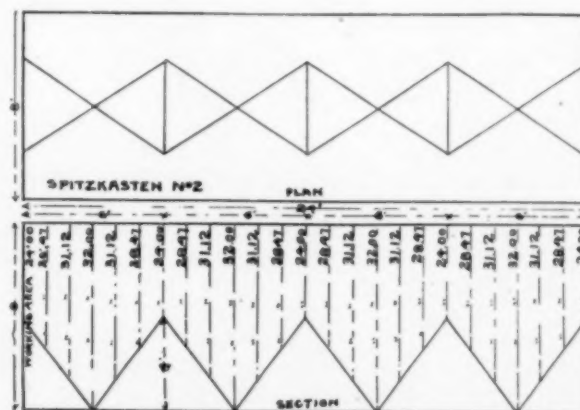


FIG. 3.—SPITZKASTEN.

any kind are created. Among these can be classed the so-called "baffle boards," which are supposed to throw the particles down to the bottom of a settling device of any kind. Also, the form of spitzkasten which, being composed of a series of inverted pyramids, have the overflow from one to the other at or very near the surface. Another is the "over and under" settling tank, containing partial partitions which are passed alternately above and below through constricted sections.

The ideal conditions for the recovery of slime material in proper form and condition for treatment is a gradually decreasing speed of flow for a distance sufficient to give from one to two hours for settlement, and that in a large body of water. These conditions are met in the form of spitzkasten illustrated in Fig. 2. It will be seen that the sectional area is progressively

increasing from the initial to the final end and every chance is given to the "elusive little fellow" to assert his superiority and pass down through the body of water to the point designed for his capture. "The form shown in Fig. 3 is very efficient. In this form it will be seen that the progressive sectional areas are increased and decreased in a series of identical fluctuations at each one of the sections, but there are two causes which would seem to assist in the action of classification and final settlement. One of these is that a portion of the contents is drawn off at each successive spigot and by just so much is the speed of the flow retarded. Another is that in the matter of the finest of the material, the settling action is so slow that it would hardly reach depth enough to be affected by the separating ridges until they reach a considerable distance from the initial or feed end. Above the dividing ridges is what might be called the effective sectional area of the moving body of water. Its sectional area in this case was $34\frac{1}{4}$ ft., or about 86 per cent of the total area figured to the apex. The spaces between the partitions can be said to contain dead water and consequently cannot be figured as effective, but merely acting as a body in which the particles which are carried to it are at rest. In the operation of this spitzkasten it was found that, when running normally, the water at the overflow end was, what might be called, perfectly clear for a depth of about 14 inches from the surface. As the overflow was very shallow and broad, the effect was that of drawing off from the surface without agitation. Compared with the spitzkasten the use of canvas plants gives inferior results. While with the latter the savings of values range from 40 to 50 per cent, the spitzkasten yields 60 to 80 per cent. In the discussion the question was asked whether baffles or burlap or some such coarse fabric would be of value in the spitzkasten. This question was answered in the affirmative, since their tendency is to arrest the slime particles and still cause no deflection or disturbance in the current. While it would not be necessary to clean such baffles very frequently under ordinary conditions, in some cases where the slimes are clayey and have a tendency to pack, they should be cleaned more often.

Copper.

A New Refining Process.—A new process is mentioned in *Metallurgie* of Jan. 8; it is being tried at the Kruglhütte, near Eisleben, of the Mansfeld Copper Co. The object is to avoid the silver loss during the bessemerizing of copper matte and to avoid also the troubles of the effect of sulphur fumes on vegetation. It has been found that in bessemerizing copper matte the loss of silver increases during the progress of the operation and reaches a maximum after sulphur and iron are removed. Only when a concentration of 79 to 80 per cent of copper has been reached, the loss of silver sets in. It is, therefore, proposed to stop with bessemerizing when the copper matte contains between 72 and 76 per cent of copper. This is then to be treated directly by electrolysis by a process of the Metallurg. Ges. in Frankfurt (German patent 160,046, Oct. 5, 1904). The sulphur is removed by electrolysis and obtained in elemental form. Silver, sulphur, nickel, etc., may be easily and completely recovered from the anode slimes. The cathode copper is claimed to be as good as any electrolytic copper. The process is being tried practically on a large scale in a plant for a yearly production of 600 tons of electrolytic copper and has been in use since the middle of February, 1907. It is claimed that the process has so far proven to be technically reliable on a large scale, and it is hoped that it will also prove to be a commercial success.

Tin.

Electric Smelting of Tin Ore.—F. Matonet gives in *Metallurgie* of March 22 an account of experiments with smelting of tin ore in the electric furnace. The ore used contained 50 per cent Sn, 7.6 Fe, 3.3 Zn, 1.9 Pb, 0.44 Cu, 7 S, etc. It was found necessary to roast the ore before putting it into the electric furnace. For

the furnace treatment an approximately neutral or slightly basic and alkaline slag is to be used. A resistance furnace must be employed. The chief advantage of electric smelting is to be found in the fact that the slag will absorb much less tin as silicate and will contain no metallic tin occluded in form of globules. The disadvantages of electric smelting are first the loss of tin as vapor and as dust, though this loss may perhaps be reduced when operating on a larger scale. The second disadvantage is the greater impurity of the tin produced. To remove lead from tin, a process similar to the Pattinson process may be employed. The tin product of the electric furnace contains about 2 or 3 per cent of lead. The tin-lead alloy is heated to the melting point (about 230° C.) and then slowly cooled to a temperature a few degrees above the melting point of the eutectics (180° C.). This temperature is maintained for some time. Pure tin crystallizes out while the heavier eutectic mixture of 30 per cent of lead and 70 per cent of tin remains back. This may be repeated several times. By treating a 2 per cent lead alloy in two refining processes, the lead is reduced to 0.2 per cent. The eutectic mixture is then treated by mixing to produce an alloy containing 10 per cent of lead, which is commercially used for cooking utensils.

Aluminium.

The European Aluminium Industry in 1907.—A review is given by J. B. C. Kershaw in the *Lond. Electrician*, March 27. In the early part of the year the demand for aluminium in Europe was in excess of the supply, a result partly due to the very high prices attained by copper and tin. The fall in the value of all metals which occurred during the closing months of 1907 has reacted unfavorably upon the aluminium industry, and the slackened demand with reduction in price of 50 per cent on the 1906 values, has marked the opening of 1908. It is probable that the whole world's output of aluminium in 1907 has approached 20,500 tons, there being 13 producing plants. The following figures from the price lists of the British Aluminium Co., in force at the commencement and end of 1907, show the extent of the fall that has occurred during the year:

	January, 1907.	November.
Ingot metal (pure).....	£200 per ton.	
	1s. 11d. per lb....	£106 per ton.
Sheet metal (per lb.)...1s. 10d.....	1s. 2½d.	
Wire	1s. 11½d.....	1s. 4d.

The Neuhausen Company, which controls the European markets, has also announced a reduction of $33\frac{1}{3}$ per cent upon its previous price, and this company is now offering the metal, for large orders, at a price of 2s. per kilogramme, or 22 cents per pound. On account of the high price of copper, aluminium has been substituted for many smaller parts of electrical apparatus and has proven satisfactory. The British Aluminium Co. are pushing steadily on with the large power development scheme on Loch Leven, and about 2,000 men have been continuously employed there during the greater part of the past year. Two new British companies have been floated. One is the Aluminium Corporation, with a capital of \$2,000,000. It has a water-power plant in course of erection in North Wales, from which 4,400 horse-power will ultimately be obtained, but it has also contracted with two power companies for the supply in bulk of 1,600 horse-power and of 4,000 kilowatts, respectively. The second new British company is the Anglo-Norwegian Aluminium Co., with a capital of \$550,000, which will erect a plant in Norway. New works are also nearly completed at Bussi in Italy, and in Switzerland.

Aluminium as Reducing Agent.—In the March issue of the *Western Chemist and Metallurgist*, W. H. Seamon discusses the use of sheet aluminium for the reduction of iron from the ferric to the ferrous state. In warm and boiling hot solutions containing hydrochloric and sulphuric acids, aluminium evolves hydrogen with great rapidity and reduces ferric salts to a ferrous condition. The author has used aluminium for the reduction of iron in his laboratory with such success that he has

abandoned the use of zinc and stannous chloride entirely. Aluminium is more than three times as active as zinc. The reduction from sulphuric solutions is perfect and more rapid than with zinc. By boiling for four or five minutes, most ores are completely reduced. If greater rapidity than this is desired the addition of a few cc of hydrochloric acid makes the reduction almost instantaneous. As a substitute for stannous chloride in the bichromate method, the use of aluminium is most satisfactory.

Miscellaneous.

Attack of Oxygen on Metals and Alloys.—In the *Chemiker Zeit.*, Jan. 4, E. Jordis reports on experiments of W. Rosenhaupt on the attack of copper, tin and zinc and copper alloys by oxygen. The apparently simple process of oxidation is shown to be quite complicated, especially with respect to the influence of temperature on oxidation. At ordinary temperature none of the metals or alloys is attacked. Copper begins to be oxidized at about 80° C., tin at 100° and zinc at 150°, but with increasing temperature the comparative effect of oxygen on the different metals changes. Certain alloys are less oxidizable than the pure metals. It is interesting to note that the property of zinc of being ductile between 100° and 150° C. does not influence its liability to oxidation.

A New High-Temperature Electric Furnace.

An extended investigation into the suitability of various types of electric furnaces for the production of carbon-free alloys has led to the development of a new furnace well suited to this class of work and for general high-temperature work. The new furnace to be described belongs to the resistance type, but the method of adjusting the resistance of the resistor and, therefore, the temperature, is as simple as it is ingenious.

It has long been known that the resistance of granular carbon can be varied within wide limits, by simply compressing it more or less. In the furnace to be described the same principle is employed, not with granular carbon, however, but with a column of carbon discs which can be screwed together to make more or less intimate contact. (See the patent of Marsh in the *Analysis of Current Electrochemical Patents*, in this issue.) By this construction the furnace attains the advantages of being simple and robust in construction, simple in operation, close in temperature regulation, economical in current consumption and having a resistor that is durable and easy and cheap to replace.

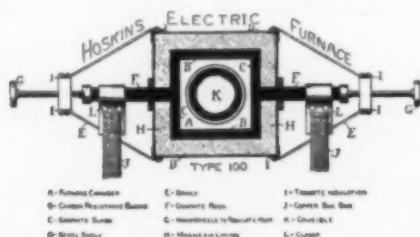


FIG. 1.—DIAGRAM OF ELECTRIC FURNACE.

The construction of a crucible furnace of this type is shown diagrammatically in Fig. 1.

A number of carbon or graphite plates are placed face to face along two sides of the furnace between end plates of graphite and with these end plates, form the walls of the furnace chamber. Graphite rods contacting with the end plates serve to both connect the resistor into circuit and to communicate pressure to the resistor plates applied by means of the screws. The resistor and end pieces are surrounded by a refractory material such as magnesite and contained in an iron casing. The current is carried to the furnace by heavy copper conductors clamped on the graphite rods.

The furnace operates best at from 10 to 50 volts, depending on its size. Where alternating current is available, a special transformer is used to reduce the line voltage to that required by the furnace. An ammeter is connected in the primary circuit as a guide to the control of the furnace. Where alternating current is not available, a rotary converter or a 10-volt plating dynamo or a storage battery may be used as source of current.

The furnace is started with little or no pressure on the resistor plates. On account of the poor contact, the resistance is high and the current small. By increasing the pressure (by simply turning the hand wheel of one of the screws), the current is brought up to the proper value, as indicated by the ammeter. The heat is developed principally at the points of contact of the resistor plates, on account of the high resistance

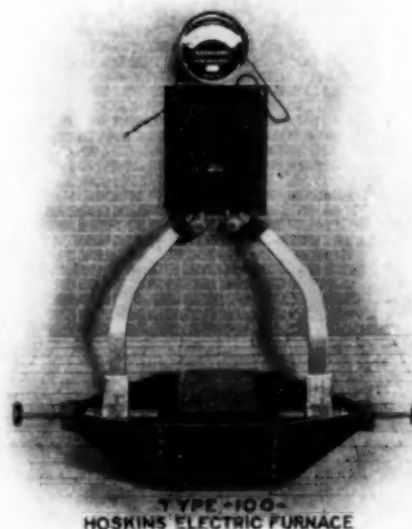


FIG. 2.—OUTSIDE VIEW OF ELECTRIC FURNACE.

at these points. The contact resistance being high compared with the resistance of solid carbon, the cross-section of the resistor can be made large and, therefore, strong and durable. The furnace can be operated efficiently, until the resistor is practically all consumed. The renewal is made by inserting the proper number of plates between the graphite end plates and is the work of a moment.

The number and thickness of the resistor plates depends upon the size of the furnace and the voltage employed.

This furnace can be operated by an unskilled person and is built to withstand hard usage.

The furnace as usually constructed will stand temperatures up to 1800 C., but can be made for a temperature of 2000 C. or above. It can be built as a tube, a muffle or a crucible furnace.

As a crucible furnace it is suitable for melting of platinum, gold, silver, nickel, cobalt, copper, iron, chromium, steel, etc.; for heating the barium chloride used in hardening steels; for determining the melting points of fireclay and similar materials, and for general experimental work with refractory materials.

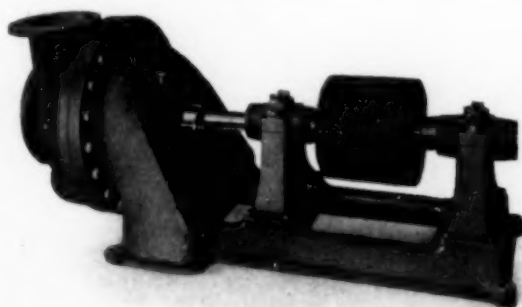
The muffle furnace will not reach the highest temperatures, being limited by the ability of the muffle to stand the heat, but will be found valuable for treating special steels, for assaying and other work requiring a temperature beyond the safe range of wire-wound furnaces.

The tube furnace will be found useful for a variety of experimental work requiring a uniform and high temperature under perfect control.

These furnaces are being placed on the market by the Hoskins Co., of Chicago, and are designated as Type 100.

Improved Rotary Pump.

The adjoining illustration shows an improved rotary pump, recently placed on the market by the Glens Falls Machine Works, of Glens Falls, N. Y. The general form of the blades is of great importance in this type of pump, because the liquid is driven through the fan, partly by the pressure of the blades on the liquid and partly by centrifugal force. The liquid is discharged with but little rotary motion, the resistance to the outward motion of the liquid being so small that the oblique action on the blades is sufficient to effect the discharge without imparting to the liquid the same speed of rotation as is given to the fan. The principal object is to effect the discharge of as large a volume of liquid as possible with the least rotary motion. The great efficiency and ease with which the pump does its work is in the impeller, which is of the enclosed type, having two veins designed on easy and efficient curves to impel the liquid with the least amount of power. There is also a projecting shoulder; either side of the impeller is finished a little larger than the diameter of the intake in the pump so as to get a close contact with a recess on either side of the shell



ROTARY PUMP.

that does not allow the liquid to be churned or re-pumped. The impeller is extra wide and is perfectly balanced against vibration and enthrust. The result is higher efficiency or more work for the same amount of power.

The suction inlet of the pump is of large area, giving a free and easy entrance to the same. It will also be seen that the bearings are long and that the pedestals are widely spaced on either side of the pulley. The pedestals are of oval shape. There are no sharp corners to come in contact with the belt, if it runs off the pulley; the bearings are especially constructed with the view of the oil not getting on the pulley and belt. The packing glands are made of bronze and are extra long, giving ample room for packing and assuring tight packing boxes without excessive pressure and friction on the revolving shaft.

The connecting parts of the pump are machine finished; all joints are planed or turned. They are always in line and require no adjusting. The pump works with an easy rotary motion. The quantity of liquid discharged in a continuous and steady stream is very large in proportion to the power expended; moreover, the pump is simple of construction and of very great durability.

Basket Anodes.

While basket anodes have been used for special purposes in electrolytic work in the past, especially for detinning tin-plate scrap, they are now finding also application in electroplating shops. In this case they are used with the intention to utilize waste metal of any form, such as brass and copper turnings, old anode butts, grain nickel, etc., directly as anode material in the plating tank. Metal in its cheapest form may always, therefore, be used without any waste.

Another advantage is that it is possible to provide easily a very large anode surface; further, that it is easy to regulate the anode surface according to the requirements of the special case or the metal under treatment. The adjoining figures show the basket anode which has been invented by the Ameri-

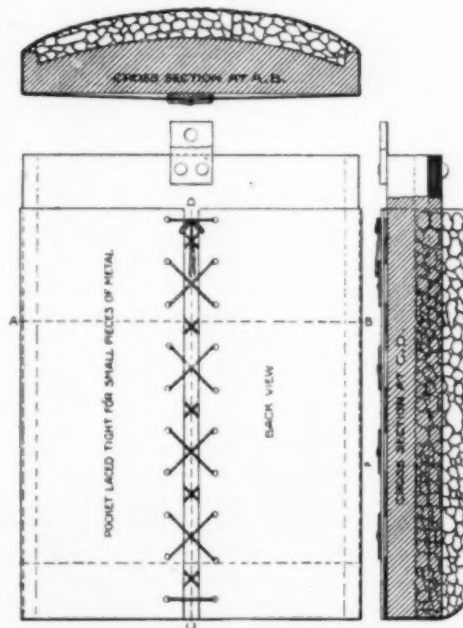


FIG. 1.—LACED BAG FOR SMALL METAL PARTICLES AND CROSS-SECTIONS.

can Nickeloid & Manufacturing Co., of Peru, Ill. They were primarily made for their own use, the company employing now some 1,400 lineal feet of the same and also introducing them elsewhere.

When employed in a vertical position the metal particles must, of course, be enclosed in a bag, and according to the size

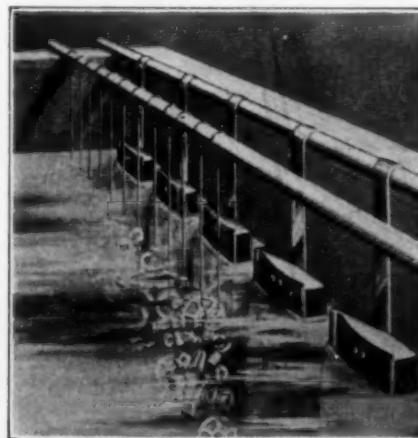


FIG. 2.—BASKET ANODES IN BATH.

of the particles the bag may be laced more or less loosely. Fig. 1 shows a snugly-laced bag for small pieces of metal. The metal may be put in the pocket of the anode with a small shovel, or if this is impracticable the slab is placed in a horizontal position, then the metal is laid on it, the burlap is put in place and laced.

Fig. 2 shows the anodes suspended in the bath.

For cleaning by electricity the anodes are used without the pocket. If the anodes suspended on the sides of the tank do

not provide sufficient surface, additional anodes may be placed in the tank in a horizontal position about three or four inches from the bottom so as to provide room for settlings underneath. No bags are required in this case.

Dry Concentrating Table and a New Electrostatic Separator.

In the paper of Mr. W. G. Swart on ore dressing, published elsewhere in this issue, his very complimentary remarks concerning the dry concentrating table of Messrs. Sutton, Steele & Steele, of Dallas, Tex., are noted. It should, therefore, be of interest to describe here briefly this concentrating table.

The Sutton, Steele & Steele dry concentrating table resembles in appearance the ordinary well-known types of wet concentrating tables, the separation being effected by the reciprocating movement of the table, while air, under slight pressure, is merely used to cushion the ore and give such mobility as to permit of the most exact separations.

The table will treat ore ranging in size from 18-mesh to 173-mesh, and finer. An ordinary suction-blower fan is used. The air enters the table from below the deck and passes through the pervious top, where it instantly expands, causing a film of air, under pressure, on the upper side of the pervious top. This air film supports and stratifies the material under treatment vertically into strata, the heaviest on the bottom, then the next heaviest, etc., and the reciprocating motion of the table causes these strata to travel and separate from each other into zones of separated material. The head motion and table support used impart not only a variable stroke, but also a slight rising and falling motion to the deck, which causes the material to travel rapidly and also takes full advantage of the frictional values of the material being treated, an important factor that does not enter into wet-table concentration.

On account of the protection afforded by the air film, one common domestic cloth will last satisfactorily for over a year on a slime table. No riffles are used on the tables, the deck being perfectly blank. By the peculiar construction of the deck the heaviest mineral being treated is caused to stop at the upper side of the table and immediately travel lengthwise of the deck, while the next heaviest is allowed to pass beyond the heaviest, etc., the lightest, or tails, passing over the mineral before taking up the motion of the table. This causes the material being treated to stratify in bands of separate mineral, which are further cleaned in their travel toward the concentrates end of the table. (This is another application of Mr. Swart's fundamental principle that one should try to coax rather than force the little particles into separation.)

Only a sufficient amount of air is used to float the material. Consequently there is no dust, while the air, being constantly supplied from below, keeps the deck load in a perfectly fluid condition, making separation by the head motion and table adjustments both rapid and thorough.

The size of the deck of the Sutton, Steele & Steele concentrating table is 5 ft. by 12 ft., the weight is 2,500 lb., the capacity from 1,000 lb. an hour of slimes to three tons per hour on 20-mesh. One-quarter to $\frac{1}{2}$ horse-power is required for the tables and from $\frac{1}{2}$ horse-power on slimes to 3 horse-power on 20-mesh material for fans.

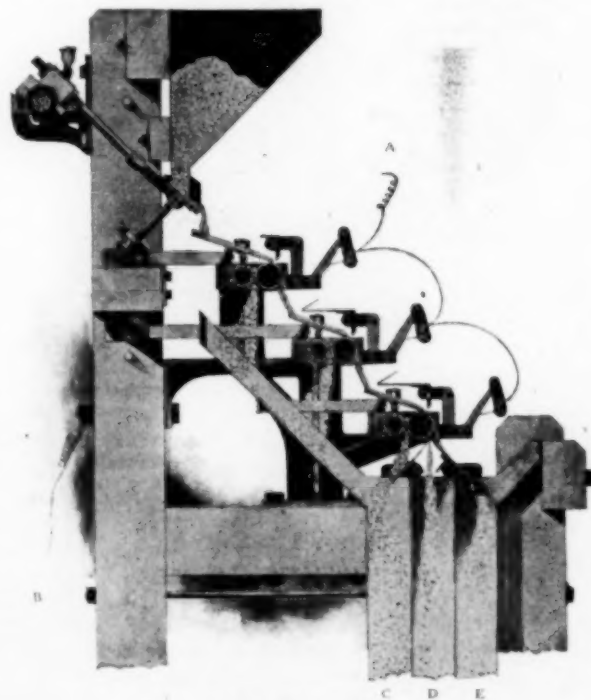
It has long been recognized that to get the best results on any concentrating machine the material to be treated must be first sized, and that the results attained by concentration depends very largely upon efficiency of the sizing end of the plant. For sizing dry material for treatment by the dry-concentrating table, Messrs. Sutton, Steele & Steele have designed a special vibromotor ore sizer. When used in connection with the dry-concentrating table it is usually arranged to deliver 10 products, viz.: On 24, 24 on 30, 30 on 36, 36 on 48, 48 on 68, 68 on 97, 97 on 129, 129 on 140, 140 on 173, and through 173 mesh, and will deliver all of the products in one operation in a perfectly

granular condition, the extreme fines going in the through 173-mesh product.

* * *

In this connection it will be interesting to describe a new electrostatic separator placed on the market by Messrs. Sutton, Steele & Steele. It is shown diagrammatically in the adjoining illustration. The separator consists essentially of a revolving metallic electrode, which is connected to the earth, and to which the commingled mass of particles is fed. Adjacent to this grounded electrode is an opposing electrode consisting of a row of sharp needle points. These points are connected to a high-tension source of electricity, preferably an electrostatic generator. The electricity delivered to the pointed electrode instantly passes off by convection to the grounded electrode, and as the opposite side of the generator from that connected to the pointed electrode is grounded, an electrical circuit is established. The electric charges supplied to the grounded electrode from the pointed electrode are dissipated as fast as received and the mass of commingled particles are thus, by the revolving of the electrode, carried through this convective discharge.

The conducting particles are not affected thereby, as resting on the grounded electrode they part with as much electricity



DIELECTRIC SEPARATOR.

as they receive, and for this reason remain inert; the dielectrics or non-conductors, on the other hand, intercept the charge, are polarized, and adhere to the grounded electrode.

The process consists in applying an alternating or interrupted charge to an electrode to which the commingled mass of particles is delivered, and of regulating the duration of the charges so applied to conform to the dielectric susceptibilities of the different particles. For instance, suppose we have two minerals, A and B; we will suppose that A can receive a charge in one second, while B will receive one in half of a second; it is evident that if the charge be alternated every half second, then A will only be half charged, the continual reversal of the charge keeping A in this condition, or at a constant difference of potential from the electrode. Consequently A adheres to the electrode, while B remains inert.

This is what the makers call "dielectric hysteretic impedance," or the lag of charge, as the residual polarity in the particle modifies the value of the applied charge, causing a lagging of the charge on the particle and consequently the adhering of the particle to the electrode. The name is rather peculiar.

As the principle of this process is based upon the specific inductive capacity of the particles, it is frequently possible to separate minerals both of which may be classed as non-conductors, but which may differ in their specific inductive capacity. Such minerals, for instance, as zinc blende and fluor-spar or barite and zinc blende. The "dielectric separator" was primarily designed for separating iron or copper sulphides from zinc blende, copper sulphides from garnet, zinc from silica, fluor-spar or barytes, garnet from mica, etc., though its field is much wider.

The standard form of the dielectric separator, as shown in the illustration, has three separating electrodes with 6-ft. feed. A is the connection to the source of high potential electricity. B is the ground connection which grounds the entire machine, except the pointed electrodes to which A is connected.

C and E can be either concentrates or tailings, depending on material being treated and the regulation of the charge, as any dielectric can be caused to adhere to the revolving electrode. For instance, when treating iron or copper sulphides and zinc blende to remove the iron or copper, the zinc is stuck to the electrode and passes into hopper C; and when treating zinc blende carrying silica, fluor-spar or barytes, the silica, fluor-spar or barytes adhere to the electrode and pass out into hopper C, while the zinc blende is not affected by the current used and passes into hopper E.

The middling product D, which is usually very small, remains about the same as the crude material, or heads.

The standard size of dielectric separator has a capacity of about 10 tons per day. It treats unroasted raw material, the power requirement being about 3 horse-power for separator and generator. In separating zinc-iron concentrates or middlings, a zinc concentrate carrying less than 1 per cent iron and an iron concentrate carrying from 1 to 2 per cent zinc are obtained.

Notes.

American Electrochemical Society.—At the March meeting of the board of directors the following gentlemen were elected members of the society: Louis E. Ward, Dow Chemical Co., Midland, Mich.; Dr. G. Baborovsky, University of Prague, Bohemia; S. T. Hansen, Jr., Missouri River Power Co., Helena, Mont. At the April meeting of the board the names of the following gentlemen will come up for election to membership: Maurice L. Dolt, Brown University, Providence, R. I.; Thomas A. Mitchell, West Virginia Paper & Pulp Co., Mechanicville, N. Y.; John T. Baker, president J. T. Baker Chemical Co., Phillipsburg, N. J.; William L. Spalding, superintendent Electrolytic Refinery, Buffalo Smelting Works, Buffalo, N. Y.; Paul R. Manahan, Avery Chemical Co., Peabody, Mass.; James M. Breckenridge, student, University of Wisconsin, Madison, Wis.; Oscar R. Foster, De La Vergne Machine Co., New York City; John A. Yunk, General Electric Co., Harrison, N. J.; E. F. Milterberger, superintendent Canada Zinc Co., Nelson, B. C.; W. C. Slade, Brown University, Providence, R. I.

The C. J. Bogue Electric Company, manufacturers of low-voltage and electroplating generators, engravers' lamps, search-lights and special electrical apparatus, will remove on May 1 from their present factory on Centre St. to more commodious quarters at 513-515 West Twenty-ninth St., New York City.

The Crocker-Wheeler Company is moving at the end of April their New York City offices to 32 Cortlandt Street in the Cortlandt Building, Hudson Terminal. This brings the New York office into close touch with the works and main office at Ampere, N. J.

The Dow Chemical Mfg. Co. of Mansfield, Ohio, have opened a sales office in Chicago, Ill., in the American Trust Building, rooms 644 and 646.

The Polytechnic Institute of Brooklyn have sent us the catalogue of the College of Engineering for 1908-9, with details as to

the courses on chemistry and chemical engineering, civil engineering, electrical engineering, mechanical engineering and a graduate course in science, etc.

The Bausch & Lomb Optical Company of Rochester, N. Y., have issued a very neat pamphlet entitled "A Triangle Alliance in Optics." As is well-known, the company is the outcome of the fusion of three celebrated concerns, the Bausch & Lomb Optical Company, the Carl Zeiss Optical Works, of Jena, and George N. Saegmuller. The booklet should be of great interest to scientists, teachers and all users of optical and scientific instruments generally.

The Denver Fire Clay Company have sent us pamphlets on their Case gasoline furnaces, Case laboratory crushers, Idler's laboratory rolls, and their clay crucibles, scorifiers and annealing cups. They also announce that their glass-blowing department is again in full operation and is prepared to do all sorts of special technical glass blowing and repair work.

The Schutte & Koerting Company, of Philadelphia, have sent us a number of illustrated interesting pamphlets on various products of their works. Among them are Schutte-Koerting rotary lead fans for chemical works, hard bronze valves, Schutte stop, check, stop check and emergency valves, balanced valves, extension rods and stands for balanced valves, balanced trap and trap throttle valves, etc.

Water-Jet Eductors.—The Schutte & Koerting Company, of Philadelphia, have sent us their recent pamphlet on Koerting water-jet eductors. Water-jet eductors are extensively used in connection with city water pressure to drain cellars, pits, cess-pools; in city water-works to wash the filter sand; in power stations and on board ship to discharge the ashes from the boiler room to a convenient location; in mining regions where high-pressure water is at disposal, to sink shafts, to empty places where it is hard to install a pump on account of insufficient space. The actuating fluid can either be taken from a local source, such as water-works, natural water sources in mining regions, water columns, or can be supplied by pressure pumps. The same pamphlet contains illustrations and descriptions of the Koerting water-jet sand-washing plants and of Koerting water-jet ash conveyors.

Welding of Pipes and Rods.—The Goldschmidt Thermit Company have just issued a pamphlet on butt-welding of wrought-iron and steel pipes and rods by the thermit process. The process is of special value for tubes for steam pressure, for deep well lining, for hydraulic work, for compressed air, for ammonia (gas or liquid), for Shelby tubes, and for rods of rectangular or circular section for reinforced concrete.

Wood Preservation.—One of the significant signs of the times is the awakening of the American people to the dangerous destruction of their forest wealth and the necessity of a wise use of what remains of it. Preservative treatment of timber, to lengthen its time of service, is therefore of extreme importance. One of the most effective preservatives is creosote, produced from coal tar, as obtained as a by-product from illuminating gas plants and by-product coke-ovens. Much study is being devoted by the United States Forest Service to creosote oil, to determine what its composition should be to give the best results in preserving timber, under different conditions, and how the most desirable creosotes may be obtained. The reports of these studies, together with detailed description of the more economical processes of applying the preservatives to wood, have been worked into circulars which the government has placed at the disposal of all users of timber and which will be furnished to all who make the request of the Forester at Washington.

The Ames-Alloy High-Pressure Sheet Packing does not melt below 750° Fahr. and has been tested by the Tight Joint Company up to 6000 lb., so that it is suitable for the highest steam pressure and hydraulic work. It is carried in stock by the U. S. Indestructible Gasket Co., of New York City, 1/32 in.

1/16 in. and 3/32 in. thick and up to 84 in. wide and is made to order from 1/64 in. to 2 in. thick for special uses. The company also furnishes gaskets in any size or shape, according to drawings, for flanges, valves, steam chests, etc., taking the place of wire, corrugated copper or steel gaskets, and particularly the many varieties of rubber and asbestos sheet and leather packings. Ames-alloy high-pressure sheet does not deteriorate, dry out or crack in the boiler room or in service. It is thought to be an ideal substitute for rubber sheet packing anywhere on land or sea, as it is oil, acid, rust and heat-proof.

Perforated Metal (steel, brass, zinc, copper, tin, etc.) is steadily gaining in application as a substitute for wire cloth and other forms of screening. The Allis-Chalmers Co. state that they are now applying it for mining screens of all kinds, in placer grizzlies for hydraulic mining, in screen plates to withstand acidulated mine water, and in separators for coal, ore phosphates and all crushed and ground minerals. The needle-slot screens used for stamp batteries, Huntington, Chilian and other mills are made of this material. For ore-dressing and concentration, perforated metal screens are used in the fixed and shaking screens for jigs, pans, agitators, washers and sizers. The screening apparatus for sluices and waterways is made from this material, and so are baffle and condenser spray plates and pipes. In glucose, sugar, starch and chemical works, perforated metal screens are used for corn shellers, degerminators, attrition mills, separators, etc. Perforated metal has replaced wire cloth for a great many of the purposes for which the latter was formerly used. It is much stronger, more uniform in size of hole or mesh and less liable to tear or rust out. In case of breakage, the damage can be easily repaired without affecting the entire sheet. It is often desirable, also, to arrange screens with certain portions blank. This can easily be done when perforated metals are used, but it is impossible with wire cloth. Unlike the wire fabric, the perforated metal presents a perfectly smooth surface, allowing the material to pass over it smoothly and quickly and it is not liable to become clogged, making it much more satisfactory for sizing, cleaning and separating.

Gas Engines for Steel Foundry.—During January, the Duquesne Steel Foundry, which operates a large plant in the Pittsburg district, decided to adopt the gas-power system to operate the works formerly driven by steam. The initial equipment will consist of a 400-hp (max.) Westinghouse gas engine of the three-cylinder vertical enclosed type, direct connected to a 240-kw generator, which will serve the various motor drives around the plant. The plant will operate on natural gas, which is available in large quantities in the Pittsburg district for power as well as lighting at such low rates as to render gas power the cheapest form of power available in this section of the country.

Asbestos Wood is an invention of Prof. Chas. L. Norton, of the Mass. Inst. of Tech., and is mineral in character, being made principally from asbestos fiber, but has the physical characteristics of ordinary wood. It is adaptable to manipulation by means of wood-working tools. As a fireproof substitute for wood, slate, marble and fiber for building, construction and electrical insulation it is now being placed on the market by the W. H. Johns-Manville Co.

Liquid Air.—The March issue of *Compressed Air* contains a detailed description of the new air-liquefying engine at Norwich, Conn., with a reproduction of 12 indicator diagrams. This is the first plant in America for making cheap liquid air, and the first ever successfully put into operation for producing liquid air by the refrigerative effect of expanding air in an air-expansion engine. The cards illustrating the description were taken when the exhaust air from the engine was in form of vapor, at from 220° to 310° below zero Fahrenheit.

"Permanized" Negative Plates for Storage Batteries.—Bulletin No. 9 of the General Storage Battery Company deals with the ingenious and strikingly simple process of Mr. J. Bijur for assuring the maintenance of capacity of negative Plate

plates. Plates treated in Mr. Bijur's method are called "permanized." We described and commented at length on the process in our April issue, 1907, our Vol. V, p. 113 and 143.

Deflocculated Graphite for Lubrication.—The International Acheson Graphite Company, of Niagara Falls, have now placed on the market their graphite, grade 1340, which is a disintegrated, extremely unctuous product of greatest purity for lubricating purposes. Mixed with oil or grease it not only improves their lubricating qualities, but gives them a "body" that prevents the parts from wearing away, as occurs when oil and grease are used alone. Deflocculated graphite is produced by reducing the highly unctuous graphite to the colloidal condition in oil or water. It then remains permanently suspended. One of the useful fields of application for this graphite is mixed with oil that is to be fed through an oil cup. The introduction of a fraction of 1 per cent. greatly reduces the oil consumption. Used in cylinder oil it increases the compression and improves the lubrication. The automobile people are especially delighted by the introduction of Mr. Acheson's "Oildag" (deflocculated Acheson graphite in oil) since it reduces the oil consumption at least 50 per cent. and keeps the engine and valves in perfect condition.

The Bristol Company, Waterbury, Conn., has come under the control of Prof. William H. Bristol, whose inventions this company has been manufacturing since it was first organized in 1889. The business, which has been carried on under the personal name of William H. Bristol, at New York, will hereafter be combined with the Bristol Company, and by this consolidation of interests the Bristol Company will now have the most complete line of recording instruments for pressure, temperature, electrical tests, and for a great variety of other applications. Among them are Bristol's pressure gauges and steel belt lacing, as well as the William H. Bristol electric pyrometers and patented smoked chart recorders, which have been described in detail in our columns. The new pyrometers have come into wide use, there being, for instance, fifty of these pyrometers in service in one of the large steel plants. Another new instrument which will soon be put on the market is the "long-distance electric thermometer," designed especially for indicating and recording refrigeration, atmospheric and drying temperatures. This instrument will fill a long-felt want for use where it is desired to quickly indicate at some central station by means of switches the temperatures at several distant points.

Mr. Edward G. Acheson of Niagara Falls, has received from the American Academy of Arts and Sciences the Rumford Medal for his splendid electric-furnace work which has been "a contribution to the direct progress of man."

Mr. Edward R. Taylor of Penn Yan, N. Y., has received the Elliott Cresson medal from the Franklin Institute in Philadelphia, for his improvements in the manufacture of carbon bisulphide and in the construction and operation of closed continuous-working electric incandescent furnaces.

Matter in the Ionized State.—We have received from Dr. Frank T. F. Stephenson, lecturer on chemistry at the Detroit College of Medicine, a reprint of his paper recently delivered before the Wayne County Medical Society, at Detroit, Mich., on "matter in the ionized state." The author discusses the subject in an interesting way with special reference to physiology and medicine.

Digest of U. S. Patents.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

CALCIUM CARBID (Continued).

681,096, August 20, 1901, Henry Spencer Blackmore, of Mount Vernon, N. Y.

Produces carbids by a reaction within an inert molten bath. Magnesium carbide and calcium fluorid are produced by feeding

magnesium fluorid and calcium carbid into a molten bath of sodium fluorid two parts and potassium fluorid one part. The source of magnesium may be a compound other than the fluorid, reducible by calcium carbid. Inert baths of molten double chlorids or other haloids may be employed. Other carbids than that of calcium may be used as reducing agents.

683,962, October 8, 1901, Hudson Maxim, of New York, N. Y.

Produces calcium carbid from lime and carbon, fed into an electric furnace consisting of a horizontal rotating tube. The current is supplied by rows of electrodes passing through the sides. The tube is a steel shell lined with fire-clay. A tubular molten lining is maintained within the clay lining by centrifugal action. This supplemental lining may consist of "any suitable substance or compound having proper electrical resistance and a specific gravity higher than that of the material to be treated," or it "may be formed or built up from and be constantly renewed by a portion of the materials under treatment, or from one of the products of the reaction effected in the furnace," for example, calcium carbid. This molten layer may act as a resistance conductor, being in contact with the exposed inner ends of the electrodes, entering the sides of the tube. The furnace is cooled by water sprayed on the outside of the tube.

675,646, June 4, 1901, John Zimmerman and Isidore S. Preiner, of Chicago, Ill.

Compresses and feeds a plastic mixture of lime, carbon and a binder, such as borax, graphite and iron, or molasses, upward into an arc, sprung between horizontal electrodes. The molten product flows downward into a receptacle beneath the furnace. When one receptacle is filled, it is shifted and another replaces it. The device for compressing and feeding the plastic mixture is a vertical or inclined tube, beneath the electrodes, containing a rotating screw conveyor. The outlet is rectangular and "of substantially the same size as the arc."

676,514, June 18, 1901, Walther Rathenau, of Berlin, Germany.

Produces pure carbid from a charge of lime and coal either of which contains silicon, by adding a metal or its oxid, preferably iron. When the carbon component is anthracite containing 25 per cent. of silica, the charge preferably consists of anthracite, 60 parts; lime, 56 parts, and metallic iron, 28 parts. The resulting silicide of iron contains from 20 to 25 per cent. of silicon.

680,050, Aug. 6, 1901, Claude Marie Joseph Limb, of Lyons, France.

Produces carbids of barium, strontium or calcium, from their sulfids or sulfates. For example, electrically smelts a mixture of barium sulfid, carbon, and a metal or metallic oxid. The product is barium carbid mixed with the corresponding metallic sulfid. Or the charge-mixture may consist of a mixture of barium sulfate, carbon and a metal or metallic oxid. The product is treated with boiling water, to decompose the carbid into caustic baryta and acetylene, this gas being collected, compressed, dissolved in acetone, or chemically utilized. The baryta is crystallized out. The small remnant of sulfid may be separated, before filtering, by adding a metallic oxid or iron sulfate. In smelting, may use either a direct current, producing electrolytic effects, or a simple or polyphase alternating current.

NEW BOOKS.

LEAD REFINING BY ELECTROLYSIS. By Anson G. Betts. 403 pages, illustrated by plates and figures. Bound in cloth. Price, \$4. New York: John Wiley & Sons.

THERMOCHEMISTRY. By Julius Thomsen. Translated from the Danish by Katharine A. Burke. 510 pages, illustrated by tables and diagrams. Bound in cloth. Price, \$2.50. New York: Longmans, Green & Co.

"The author is emeritus professor of chemistry in the University of Copenhagen. The experimental work for this book was carried out in the years of 1851 to 1885, but the greater part belongs to the last twenty years of that period, during which the author was director of the chemical laboratory of the university. His information was published in a four-volume work, "Thermochemische Untersuchungen." He now wishes to make the original results more accessible than in the larger work. He reviews the whole of the numerical and theoretical results without devoting much space to experimental details. He has reduced the work one-fifth and has provided easy access to the results themselves."

THERMODYNAMICS OF TECHNICAL GAS-REACTIONS. By F. Haber. Being seven lectures, translated by Arthur B. Lamb. 375 pages. Bound in cloth. Price, \$3 net. New York: Longmans, Green & Co.

LABORATORY EXERCISES IN PHYSICAL CHEMISTRY. By F. H. Getman. Second edition, revised. 295 pages, illustrated by tables. Bound in cloth. Price, \$2 net. New York: John Wiley & Sons.

THE BLAST FURNACE AND THE MANUFACTURE OF PIG IRON. An elementary treatise for the use of the metallurgical student and the furnaceman. By Rob Forsythe. 368 pages, illustrated by diagrams. Bound in cloth. Price, \$3. New York: David Williams Co.

IRON AND STEEL. By J. H. Stansbie. 388 pages, illustrated. Price, \$2 net. New York: D. Van Nostrand Co.

"This book aims to give as comprehensive a view as its limits will permit of the modern aspects of iron and steel manufacture, together with a sufficient account of its history to enable the reader to follow its march of progress. The main portion has been written from notes used for the courses of lectures and has been brought to date by references to the latest books."

SIMPLE MINE ACCOUNTING. By D. Wallace. 63 pages. Bound in cloth. Price, \$1. New York: Hill Publishing Co.

LIQUID AND GASEOUS FUELS, and the part they play in modern power production. By Vivian B. Lewes. 448 pages, illustrated by diagrams and tables. Bound in cloth. Price, \$2 net. New York: D. Van Nostrand Co.

TOWN GAS AND ITS USE; for the production of light, heat and motive power. By W. H. Y. Webber. 281 pages, illustrated. Bound in cloth. Price, \$2 net. New York: D. Van Nostrand Co.

COAL. By Ja. Tonge. 281 pages, illustrated. Bound in cloth. Price, \$2 net. New York: D. Van Nostrand Co.

INDIA RUBBER AND ITS MANUFACTURE; with chapters on gutta-percha and balata. By Hubert L. Terry. 302 pages, illustrated. Price, \$2 net. New York: D. Van Nostrand Co.

"All information has been brought to latest date. The volume is chiefly designed for the general reader and for the technologist in various branches of industry; it has not gone sufficiently into detail to be a working handbook for the manufacture of india-rubber goods in which various problems still await satisfactory solution."

WOOD. A manual of the natural history and industrial applications of the timbers of commerce. By G. S. Boulger. Second revised and enlarged edition. 367 pages, 91 illustrations. Bound in cloth. Price, \$4.20. New York: Longmans, Green & Co.

HYDRAULICS. By F. C. Lea. For engineering students. 548 pages, illustrated by diagrams. Bound in cloth. Price, \$5 net. New York: Longmans, Green & Co.

THE STEAM TURBINE. By Rob. M. Neilson. Fourth edition, revised and enlarged. 630 pages, illustrated by numerous figures and diagrams. Bound in cloth. Price, \$4.20 net. New York: Longmans, Green & Co.

BOOK REVIEW.

PRACTICAL METHODS FOR THE IRON AND STEEL WORKS CHEMIST.

By J. K. Heess, Ph.C. 60 pages. Price, \$1 net. Easton, Pa.: The Chemical Publishing Co.; also New York: McGraw Publishing Co.

This useful little book begins with hints as to location, plan and construction of the iron and steel laboratory. Then follow several pages of general information on experiments, special apparatus, books, standard samples, and standard solutions and reagents.

The bulk of the book is devoted to the sampling and analysis of iron ores, coke and coal, limestone, slag, iron and steel, ferro-manganese, ferro-silicon, refractory materials, Portland cement, water for boiler purposes, cylinder oils, gas analysis, bearing metals. The methods are briefly stated and are, in the main, not different from those usually given in similar works on iron analysis.

THE COPPER HANDBOOK. A manual of the copper industry of the world. By Horace J. Stevens. Vol. VII: 1907. 1228 pages, bound in cloth, price \$5.00. Houghton, Michigan: Horace J. Stevens.

The endeavor of the author and publisher to improve continually on the general usefulness of this well-known yearly handbook of the copper industry is indicated by the increase in size of the volume. Volume 5 had 882 pages, volume 6 had 1116 pages and the present edition has 1228 pages.

The bulk of the volume is still the statistics of the copper mines and companies of the world with concise descriptive notes. Exceedingly useful information on about 5,000 copper companies is here given. This part of the book is, however, identical with the corresponding part in Vol. VI. The handbook is the result of the work of a single man who collects the data, critically reviews them and writes and publishes the book. It is explained that fire, sickness and loss of five months' time prevented the thorough revision of this part of the book.

However, the introductory explanatory chapters have been revised and greatly enlarged. To the old chapters on history, geology, chemistry and mineralogy of copper and a glossary of mining terms, there have been added new chapters on copper mining, milling and concentrating, hydrometallurgy, pyrometallurgy, electrometallurgy, alloys of copper, brands and grades of copper, and substitutes of copper.

These new chapters are written in plain language so as to be easily understood and are not intended for technical men, but rather for the layman, whether miner or investor. For this reason they should strongly appeal to the general public, as far as it is interested in copper, and should win many new friends for the Copper Handbook.

THE ENGINEERING INDEX ANNUAL FOR 1907. Compiled from the Engineering Index, published monthly in the *Engineering Magazine* during 1907. Bound in cloth, 435 pages, price \$2.00. New York: The *Engineering Magazine*.

For a long series of years the *Engineering Magazine* has published every month a classified list of titles of articles appearing in current issues of technical and engineering journals in this country and abroad. The present volume is a reprint of the lists published during 1907.

The list is classified to a certain extent, the main divisions being civil engineering, electrical engineering, industrial economy, marine and naval engineering, mechanical engineering, mining and metallurgy, railway engineering, street and electric railways.

The chapters on industrial economy, marine and naval engineering, street and electric railways are not classified, all items being arranged alphabetically. All other chapters are divided again into subclasses; for instance, mining and metallurgy into coal and coke, copper, gold and silver, iron and steel, mining, miscellany.

FIRST PRINCIPLES OF CHEMISTRY. By R. B. Brownlee, W. J. Hancock, R. W. Fuller, M. D. Sohon and J. E. Whitsit. 12 mo, 420 pages, 121 illustrations. Price, \$1.25 net. Boston, Mass.: Allyn & Bacon.

This first presentation of chemistry is an innovation in chemical text-books. It was prepared by five high-school teachers after two years of consultation with each other and polishing up of the material collected. It is the most sane and sensible presentation of the subject which these five experienced teachers

could collectively produce. They cast precedents and traditions aside, and going straight to the root and kernel of the matter, wrote as they would explain, in class, and omitted everything not directly and profitably leading towards their goal. Each criticized and reviewed the work of all the others, with the result that the book is very good. The result is the most lucid, the most translucent, the most careful presentation of the elementary facts and principles of chemistry to be found in English.

The order of development is chronological and rational. Combining weights are explained before atomic weights, as they should be. The atomic hypothesis is most excellently presented, not as a dictum, but as a reasonable explanation of the facts. The phenomena of catalysis, electrolysis, etc., are expressed and explained in absolutely unobjectionable terms, entirely free from hypothesis and the ordinary lingo of the theorist, instilling the jargon of his pet theory into young, receptive minds along with the first elementary facts. Such teaching as we have in this book will make strong, wide-thinking chemists, and not mere parrot-like repeaters of language which they do not and cannot understand.

There are a few slips and a few places where the emphasis might have been changed with advantage. In the difficult matter of explaining Avogadro's hypothesis (p. 74), the weights of equal volumes of elementary gases and their reacting weights are confused. Since all of the students have learned weights and measures primarily in pounds or ounces and feet, it would not confuse them to state that an ounce molecule of any gas has a volume of 22.22 cu. ft. at standard conditions. Such an appeal to the measures they are most familiar with would make the metric measures more real and better understood. In the production of sodium, Castner's name should not be omitted from the description of his apparatus; the reader would think it was Davy's apparatus which was being described. The explanation of hardening and tempering steel is wrong.

The chapter on "Solutions" is far below the level of the other chapters in disingenuousness. It pleads too much the theory of its writer. It is not so fair, so judicial as the others. It seeks to instal the theory rather than to deduce an explanation from the facts. It makes such questionable statements as "Nearly all chemical reactions require the presence of water" and "Non-electrolytes show very little activity." We will not discuss these statements; they are so far from being true that they are poor material to drop into the beginner's mind. This chapter is the poorest in the book, and this just because the theory of its writer is put in the foreground, and facts twisted and manipulated to bolster up the theory. The teacher using the book should carefully lead his students through this pit-fall.

As a general criticism on the whole book, we would say that the quantitative conception of the energy of chemical combination and reaction, as set forth by thermochemistry, is introduced too late and too little into the book. This is the one saving grace which converts chemistry from a qualitative into an exact science, and it ought to be made prominent *ab initio*. One could then explain from the beginning why hydrogen and oxygen explode, why carbon makes a hot fire, why sodium decomposes water, why carbon reduces litharge, and why many other things take place, and where the energy necessary to make them go comes from. These are too valuable assistants to be left out, as they are from most of the book. The combination of Ditte's plan in his well-known French text-book of chemistry, with the lucidity, logic and splendid clearness of this work, would make an ideal work unapproached heretofore in any language.

We hope that many new editions of this book will be required and that the authors will improve it every time in the respects indicated; we will then have an elementary text-book of chemistry which will be an ornament to American scientific literature, a boon to chemical science, and a maker of "manly" chemists.